THE CLEANING OF COAL

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CONTENTS

INTRODUCTION

BY PROF. R. V. WHEELER, D.SC., F.I.C.

THE process of cleaning coal, or the separation of dirt from it, begins in the mine. It begins with the choice of the seams to be worked and continues with the choice of the methods of working them. The care taken by the miner in separating the coal from dirt during loading at the face also affects the quality of the "raw material" delivered at the surface, and thus (sometimes to a considerable extent) determines the extent to which manual or mechanical means must be employed at the surface to purify this raw material. It is usual to confine the term "coal-cleaning" to these operations at the surface.

In the early days of the British mining industry, hand-picking at the colliery screening plant (care being taken to mine only the better seams) was all that was necessary for the production of the larger sizes of saleable coal. With the development of the coking industry, clean small coal acquired a greater value and mechanical cleaning processes were first introduced to deal with this small coal for coking purposes. Since these early days, it has gradually been realised that the gradation of the coal as mined according to size, and the removal from each size of as much as possible of the associated incombustible matter, so far benefits the consumer, in every industry, as to induce him to pay a higher price for the sized, cleaned commodity. The cleaning of coal, indeed, provides the most direct method of increasing the usefulness, and therefore the value, of the raw material and should be regarded as a most powerful weapon in the hands of the coal-mining industry, for individual collieries to meet competition, and for the export trade as a whole, to stimulate the falling demand for British coal and to recapture lost markets.

In Continental Europe and in America, as well as in Great Britain, there is a growing tendency to discard raw coal as a heating agent and to use instead an artificial fuel, coke, prepared from it. Eventually, no doubt, satisfactory processes for the low-temperature carbonization of coal will provide the domestic market with a smokeless semi-coke. For the preparation of these artificial fuels, whether gasworks coke, metallurgical coke, or the semi-coke of low temperature carbonisation, a clean coal is essential, since any dirt in the raw material is concentrated in the finished product.

At present, the mechanical cleaning of coal is practised more widely on the Continent than in Great Britain, no doubt mainly because British coals are, in general, less friable than the Continental,

and yield a higher proportion of lump coal, which can be cleaned by hand-picking. Nevertheless, there is evidence that part, at least, of the British export trade has been lost by reason of its irregular quality. Professor Henry Louis, for example, (*Journ.* Soc. Chem. Ind., 1927, 46, 545) has instanced the loss to Germany of certain Scandinavian railway contracts, not, it is stated, on account of cost, but because the German coal was better cleaned. Yet British coals have better inherent qualities, and afford a wider range of usefulness, than any in the world.

South Wales anthracite is rightly regarded as the purest anthracite mined. South Wales steam coals are unequalled. South Wales coking coals yield foundry cokes of the highest quality and, if coked in modern ovens, would produce blast-furnace cokes equal in value, if not superior, to Westphalian cokes. In South Yorkshire, the hards of the Barnsley seam provide the world's best bunker and locomotive fuel, withstanding rough handling and burning without clinker trouble. The hard steam coals of Scotland and of Northumberland have proved their worth. Durham coals, though usually classed as gas-coals, are all coking coals and in by-product practice yield dense, hard cokes, which have long been recognised as of superior quality for blast-furnace use. The Midlands supply furnace and gas-producer coals of the highest value.

With these natural resources, Great Britain has, in the past, been able to succeed in competition with foreign producers who, to gain an advantage, have been compelled to resort to mechanical cleaning. Some of the seams, however, on which the reputation of British coals has been built are no longer available in sufficient quantity to meet an extensive demand. For example, the famous Brockwell seam, which first established the reputation of Durham coking coals; the Main seam, one of the first Newcastle coals to be sent to the London market; the Rhondda No. 2 and Rhondda No. 3 seams, two of the best of the South Wales coking coals; the Trencherbone and the Arley seams of Lancashire, and the Silkstone seam of Yorkshire can no longer be mined so readily as in the past. Although there are many seams, other than these, having a similar range of properties, available in Great Britain in large quantities, they often cannot be brought to the surface in such a clean condition as could the older seams.

The time thus seems ripe for British coal producers to take a leaf from the book of their foreign competitors, to adopt extensively the mechanical cleaning of coal which has enabled those foreign competitors to discount the inherent excellence of British coals. In such a project Great Britain would once more have an advantage over the Continent, for the British coals in general contain less unremovable ash and are brought to the surface mixed with less free dirt. To produce a coal for the market with a moderate ash-content, therefore, less of the pit's output need be rejected as refuse. Moreover, the product from British mines contains less material

intermediate in character between coal and shale ("middlings") than does the Continental, so that any process of cleaning employed is more efficient and the product is purer.

It is a common belief that the cleaning of coal (and the allied art of ore dressing) originated in Germany and was developed there. This is an entirely erroneous belief. It is true that coal washing is more widely practised in Germany than elsewhere, and that two of the most successful types of washers, the Lührig and the Baum, were of German origin, but in design and operation, and in the invention of new processes, Germany in no way surpasses other countries. In the early development of the jig-washer, a Cornishman, Petherick, was responsible for important improvements; in particular, for the device of using a fixed screen and causing the water to move as a current through the material on the screen. The continuously operated jig was the invention of a Frenchman, Bérard, and the next step, the use of a false bed of feldspar, was known as the "English method" from its use in English ore-dressing practice. The two most important upward-current washers, the Robinson and the Draper, are both British inventions, as are two of the best known forms of trough washers, the Blackett and the Elliott. Although but few improvements in concentrating tables have originated in this country, the development of the modern table dates from the invention of the endless belt by Brunton, of Glamorgan, in 1844, and the use of a transverse water current on a stationary surface, to remove impurities from ores, by his father in 1841. The separation of gangue from ore by differential wetting was the discovery of an Englishman, Haynes, whilst froth flotation was first employed by Elmore, also an Englishman. The dry-cleaning of ores and of coal is an American invention, whilst British research has given stimulus to such processes, resulting in the devising of two methods for the pneumatic treatment of unsized coal, the Kirkup and the Raw.

The authors estimate that only about one-quarter of the total coal brought to the surface in Great Britain is subjected to mechanical cleaning processes. There is therefore plenty of scope for the development of the practice of coal-cleaning, a development which, it is not too much to hope, would restore to Great Britain, still the leading coal-exporting country in the world, her former markets. With increase in the practice of coal-cleaning, no doubt the development of old processes and the designing of new, in which British invention has in the past played so large a part, will go hand in hand. The authors, by collecting together present knowledge of the art of coal-cleaning and discussing the merits and demerits of the various processes, have made an important contribution to technical literature, which should aid considerably in the development foreshadowed, by suggesting lines of thought to the inventive genius of British engineers.

THE CLEANING OF COAL

CHAPTER I

THE IMPURITIES IN RAW COAL

THE object of coal cleaning is the separation of pit-coal into two or more products, one of which is a more marketable commodity than the original coal. This is accomplished by removing most of the dirt * or incombustible matter associated with the coal when it leaves the pit.

For purposes of coal cleaning, the dirt associated with raw coal may be considered as "fixed" dirt and "free" dirt. The fixed dirt was more or less intimately intermingled with coal during the process of coal formation, but the free dirt is mechanically mixed with the coal as a result of mining operations rather than as a result of natural processes. The fixed dirt in coal originated partly from the inorganic constituents of the original coal-forming plants and partly from adventitious mineral matter which became intermingled with the plant *débris* during the process of coal formation. To appreciate fully the nature of the fixed dirt (and also of the free dirt from roof, floor, dirt partings or pyritic inclusions) it is necessary to consider the origin of coal and the manner in which coal seams were laid down.

Coal is defined by Stopes and Wheeler as "a compact stratified mass of 'mummified' plants (which have in part suffered arrested decay to varying degrees of completeness), free from all save a very low percentage of other matter." They continue: "Veins, partings, etc., which are found in nearly all coals, are local impurities and are not part of the coal itself."

The plants from which were formed most of the bituminous and anthracitic coals of Europe and the eastern portion of North America, grew during the Carboniferous period, 150 to 190 million years ago. The principal coals of China, India, Australia and South Africa, and coals of the Sarre district, and of Saxony, originated in the Permo-Carboniferous or the Permian period, 70 to 150 million years ago. Those found in the western part of North America, in Japan, and in New Zealand, date from the Eocene period, 10 to 20 million years ago (Lewis, *Fuel*, 1922, 1, 76).

It is believed that the great coalfields of the Carboniferous system originated in a luxuriant, monster vegetation which flourished, in

* Dirt is matter out of place.

a tropical or sub-tropical climate, in a rich virgin soil with large areas of inland sea and fresh water lakes. The early plants reproduced through the medium of soft spores which were released in water, were distributed by water and could only germinate under water. The first flora (vascular cryptogams) consisted of great tree ferns, gigantic equisetums (horsetails), cycad ferns and lycopods. Many of these plants took the form of huge-stemmed trees, some over 100 ft. high. The lycopods (Lepidodendron and Sigillaria) were dominant in many coal basins, but disappeared very early in the history of land plants, Lepidodendra not even reaching the Permian period.

Probably most of Great Britain was covered by forest growths, and it is generally believed that the coal seams in the major coalfields resulted from the growth and accumulation of plant *débris in situ*, modified, in part, by drift accumulation. The coal in some of the minor coalfields and in isolated seams resulted, apparently, from drift material, which would be associated with larger quantities of mineral detritus than were the *in situ* growths forming the major coalfields.

In the course of time the coal-forming plants were subjected to mouldering decay, and the process of coalification of the decaying mass proceeded for thousands of years. During this time the coal-forming mass was penetrated by water- and air-borne mineral detritus, which became more or less intimately intermingled with the plant *débris*. At certain periods also the deposits may have been covered with sediments brought into the inland seas and lakes by rivers, or as a result of the lowering of the level of the forest growths. These sediments formed the dirt partings and roofs of coal seams.*

As the coal-forming material decayed it gradually hardened, and vertical cleavages (or cleats) were formed. In these cracks, percolating water containing calcium, magnesium and ferrous salts deposited the white partings of mineral, sometimes called ankerites. Ferrous sulphate in percolating waters reacted with the decaying vegetable matter to form iron pyrites. If the reduction of ferrous sulphate took place in the early stages of coal formation, the deposit of iron pyrites would occur in small crystals throughout the coal-forming mass, and at later stages, when the mass was less permeable to percolating waters, pyrites would be formed in thin sheets in cleat partings or in lenticular masses on the bedding planes. Gypsum (calcium sulphate) is sometimes found in thin sheets in the cleat, and also, in small quantities, distributed intimately throughout the coal.

* E. McKenzie Taylor (*Fuel*. 1926, 8, 195) has shown that in salt water some of the calcium and magnesium of clay may be replaced by sodium to form a sodium-adsorption complex, which is stable in the presence of excess of neutral sodium salt, but dissociates in fresh water, giving an alkaline solution. The alkaline sodium-adsorption complex formed is impermeable to gases and liquids, and, hence, clay sediment settling on the coal-forming mass in brackish water would not mix with the plant material, but would form a protective covering.

The organic matter of pure coal is a mixture of ulmins and plant remains. It is composed mostly of carbon, hydrogen and oxygen in complex groupings. Nitrogen, from protein matter, and sulphur occur as essential parts of the molecular groupings. A small quantity of phosphorus (possibly from cell nuclei) also forms a part of the pure coal. In banded bituminous coals, distinctions may be drawn between the macroscopic appearance of different bands. Stopes (*Proc.* Roy. Soc., 1919, B, 90, 470) has called the different bands found in British bituminous coal: fusain, durain, clarain and vitrain. Fusain is the dull powdery material occurring in wedges and patches on the bedding planes; durain, the dull hard coal; clarain, the bright soft coal; and vitrain, the brilliant vitreous coal which occurs usually in thin bands in both durain and clarain.

Bright coal usually gives a reddish-brown ash, since it contains a high percentage of iron oxide (10 to 40 per cent.). The iron is mostly present in the original coal as finely-divided (microscopic) pyrites intermingled with the coal substance. Dull coal, on the other hand, gives a white ash which only contains 1 or 2 per cent. of iron oxide. Usually, therefore, only small quantities of iron pyrites are distributed throughout the substance of the dull portion of coal.*

The types of impurity found in a coal seam may be classified as follows:—

1. The residual inorganic constituents of the coal-forming plants.
2. Detrital matter washed or blown into the coal-forming mass and becoming intermingled with the plant *débris*.
3. Finely-divided pyrites resulting from interaction of the coal-forming mass and percolating waters containing ferrous sulphate.
4. Sedimentary deposits forming partings.
5. Massive deposits (pyrites) formed by decomposition on bedding planes and in the cleat.
6. Saline deposits mainly from percolating water (ankerites).

Fraction (1) is the *inherent* mineral matter of coal; fractions (2) and (3) the *adventitious* mineral matter. The combined fractions (1), (2) and (3) give the *fixed ash* of coal, which is the limiting ash content to which coal can be cleaned. Fractions (4) and (5) contribute mostly the *free dirt* in coal, that is, the mineral matter which is not intimately associated with the coal substance, but which is mixed with the coal during mining operations, and is the portion of the raw coal which may be normally removed by coal-cleaning

* Dull coal also contains layers of mineral matter (which did not originate from the inorganic matter of the plant) which are not found to the same extent in bright coals. It seems possible, therefore, that the water floods which carried the spores (and mineral detritus) into the material forming the dull coal removed many of the ferruginous salts in solution. Bright coal may have been laid down under comparatively stagnant conditions in which flooding was unusual and iron salts might then assume greater concentrations.

operations. Fraction (6) is most frequently found in laminæ on large lumps of coal, but, on screening and general handling, the coal is fractured and these plates of impurity are generally removed. In some coals (Lancashire, for example) it is of fairly frequent occurrence, but is rare in others.

Inherent Ash.—Little is known about the absolute amount of inherent ash (residual plant ash) present in lumps of "pure" coal, for it is intermingled with finely-divided mineral matter from extraneous sources. It would, however, seem likely that the "inherent ash" of coal may often be less than 1 per cent., and is seldom greater than 2 per cent. A separation of the adventitious from the inherent ash of coal was attempted by Mott and Wheeler (*Fuel*, 1926, 5, 416). Coal samples were finely ground in water and the purest coal, probably containing only the inherent plant ash, was collected by flocculation with paraffin oil, the adventitious mineral matter remaining in suspension. A series of the clarain and durain portions of the Parkgate Seam (Yorkshire), taken from different pits, was used; the average analyses of the original and purified coals are recorded in Table 1.

TABLE 1.—SEPARATION OF THE INHERENT AND ADVENTITIOUS ASH OF COAL

	Ash per cent. in Coal.			Mineral Matter removed per cent. on Total Ash.	Ash per cent. of Purified Coal. (Calc. Iron Free.)
	Before Treatment.	After Treatment.	Removed.		
Clarain .	2·20	1·56	0·64	29·0	0·91
Durain .	3·68	1·49	2·19	60·0	1·13

Complete analyses of the inherent and adventitious ashes showed them to be quite different in type. As all but a negligible amount of the iron present in coal is of adventitious origin, the results of the analyses in Table 2 are calculated on an iron-free basis. The analyses are compared with those for the middle dirt band, which separates two portions of the Parkgate seam, and a sample of seat earth of the same seam.

It will be apparent that these three kinds of impurity in coal, namely, inherent ash, adventitious ash and free dirt, are of different types, and therefore probably of different origin. Silica and alumina together form the greater part of ordinary coal ashes and are chiefly of extraneous origin (adventitious ash or free dirt). In the wood and leaves of modern flowering plants, lime forms from one-half to three-quarters of the total ash content, but the amount

TABLE 2.—COMPOSITION OF INHERENT AND ADVENTITIOUS ASHES OF PARKGATE SEAM

"Inherent Ash."

Seam Sample No.	Clarain.			Durain.		
	VIII.	V.	IX.	VIII.	V.	IX.
Ash per cent. . .	0·73	2·07	1·12	1·03	1·62	2·07
SiO_2 per cent. . .	54·1	53·4	55·8	50·9	49·6	49·8
Al_2O_3 per cent. .	38·1	38·8	42·1	42·4	43·9	46·3
CaO per cent. . .	3·2	4·2	1·1	1·1	3·6	1·1
MgO per cent. . .	1·1	0·7	0·2	0·2	0·4	0·2
Ratio SiO_2/Al_2O_3 .	1·42	1·38	1·32	1·20	1·13	1·08

"Adventitious Ash"

	Clarain.			Durain.		
	VIII.	V.	IX.	VIII.	V.	IX.
SiO_2 per cent. . .	49·8	50·8	51·3	57·0	61·4	62·1
Al_2O_3 per cent. .	42·9	45·3	46·7	40·7	35·3	36·1
CaO per cent. . .	0·1	3·6	1·0	0·3	0·8	0·9
MgO per cent. . .	trace	0·8	0·8	trace	0·1	0·3
Ratio SiO_2/Al_2O_3 .	1·16	1·12	1·10	1·40	1·74	1·72

"Free Dirt"

	Parkgate VIII. Middle Dirt Band.	Parkgate IX. Seat Earth.
Combined water per cent. .	10·1	7·6
Ash per cent. . . .	75·3	81·2
SiO_2 per cent. . . .	55·7	63·4
Al_2O_3 per cent. . . .	33·7	30·7
CaO per cent. . . .	0·6	0·4
MgO per cent. . . .	1·3	0·7
Ratio SiO_2/Al_2O_3 . . .	1·56	2·35

of lime usually found in the ash of coal is low. Other elements which are found are manganese, nickel, zinc, copper, titanium, vanadium, arsenic, phosphorus, sulphur and chlorine.

The average ash content of modern plants lies between 0·5 and 1·5 per cent. (being the highest in leaves and the lowest in bark), but it is possible that, under the conditions of more rapid growth of the Carboniferous period, the ash contents of the flora were lower. All the plant ash may not be present in coal, for some of it may have been converted into soluble salts by the humic acid (ulmins) formed, and leached out. The low ash content of South Wales anthracites is sometimes commented on, but this may only mean that the amount of adventitious ash present is low. For example, the average ash content of five anthracites was 1·26 per cent., the average SiO_2/Al_2O_3 ratio being only 0·91 (Grounds, *Fuel*, 1923, 2, 13). In general, coals of very low ash content have low SiO_2/Al_2O_3 ratios, or, in other words, have high percentages of alumina, as have the modern prototypes of the coal-forming plants.

Adventitious Ash.—The adventitious ash (or that part of it which is due to the mineral matter washed or blown into the coal-forming mass during an early stage of its formation) is not uniformly distributed throughout the coal material, but occurs usually in laminæ on the bedding planes. This is shown in the X-ray photograph, Fig. 1 (Kemp and Thomson). The adventitious ash may be seen in thin layers on several of the bedding planes. On crushing, breakage occurs to the greatest extent along the bedding planes (where the cohesion of the material is the lowest), and some of the dirtier bands will be freed from the cleaner material. In many Indian coals the adventitious ash content is so high that little purification of the coals would be possible by washing unless it were practicable to grind the coals very finely before washing. Similarly, Rhode Island (U.S.A.) anthracites were said to be unusable until they were very finely ground and cleaned. Many American coals contain "bone" particles which have a high percentage of adventitious ash. In bone coal, however, the ash is fairly uniformly distributed through the mass and little improvement by crushing and washing is possible.

Ankerites.—Ankerites occur in the form of plates, varying in thickness from ⅛ in. to thin films in the cleat. Sinnatt, Grounds and Bailey (*Journ.* Soc. Chem. Ind., 1921, 60, 17) suggest that they were formed by crystallisation from percolating saline waters which may have travelled horizontally in the seam through layers of porous fusain. They record analyses of ankerites from a number of Lancashire seams, and show that, in general, the ankerites contained over 50 per cent. of calcium carbonate, the remainder consisting mostly of magnesium and ferrous carbonates. In one 1,000 gm. sample of coal about 3 per cent. of ankerite was

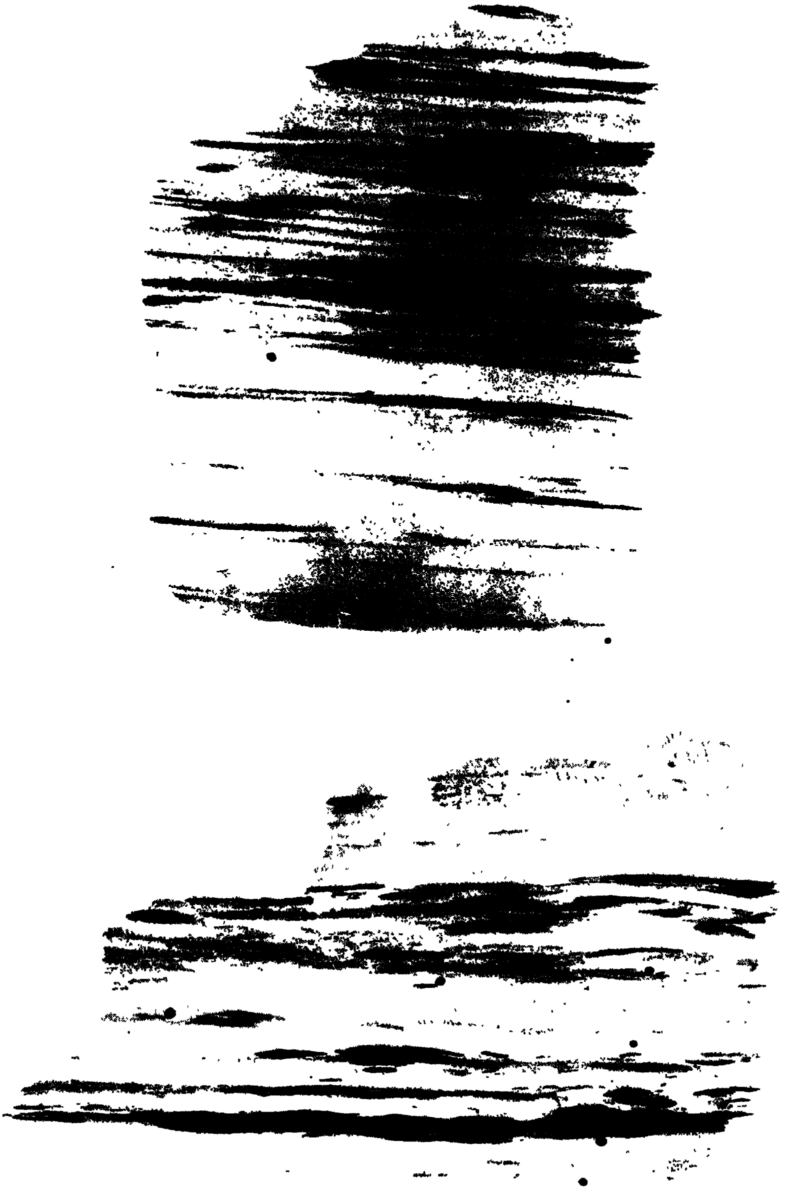

FIG. 1.—X-Ray Photographs of Coal.

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present. The fixed ash of the coal (freed from ankerites) was 2·5 per cent., and the ash contributed by the ankerites raised the total ash content by 1·7 to 4·2 per cent. The high percentage of lime present in ankerites may sometimes be the chief contributor of the lime reported in coal ash analyses, and the ferrous iron of ankerites will often account for all the iron not present as pyrites.

Dirt Partings.—Coal which is commercially workable occurs in the earth in seams which vary in thickness from about 12 in. to 12 ft., or, in rare cases, even more, but most of the coal wrought in Great Britain is won from seams of 2 ft. 6 in. to 5 ft. 6 in. thickness. A coal seam may consist of solid coal between the roof and floor, or may have dirt partings of variable thickness separating bands of coal. The roof of a coal seam may be composed of coarse-grained rock (sandstone), or of fine-grained bind (silty mudstone), or of intermediate types of material of varying grain size (sandy mudstone). The term " bind " is sometimes used indiscriminately for either mudstone or shale. The grains of varying size are called, geologically, sand, silt or mud : a sedimentary rock containing over 50 per cent. of fine or medium sand is a rock or sandstone ; one with over 50 per cent. of silt, and the remainder mainly mud, is described as stone bind or sandy mudstone ; and one with 70 to 80 per cent. of the mud grade, and the remainder mostly fine silt, is called a bind, mudstone or shale, according to the perfection of the lamination. In mining practice, sedimentary rocks in which the individual sand grains are clearly visible are termed sandstones or rock ; those rocks in which thin laminæ of fine sand, or mica flakes, and of more muddy material alternate frequently, or which are rough to the touch, are called stone bind ; and those rocks in which few mica flakes occur and which are smooth to the touch are called bind, or, if well laminated, shale (" Sections of Strata of the Coal Measures of Yorkshire," Sheffield,

The arrangement and shape of the mineral particles have important effects on the character and behaviour of a sedimentary rock. In sandstones, where the quartz, felspar and mica grains are evenly distributed, the rock is massive (often a freestone) ; but when flat mica flakes have been deposited discontinuously the rock breaks into thin slabs and is called a flagstone. In the finer-grained rocks the laminations become more marked as the proportion of micaceous minerals increases and as the ruling size of grains decreases. A bind has a relatively small proportion of micaceous substance, but shale has a much larger proportion of mica resulting from the weathering of felspar. A sandstone forms a good roof, which is not liable to fall during the general working of a seam, but laminated shales give trouble by falling to a certain extent when the coal is removed.

The floor of a seam, or the under clay, is called clunch, spavin or seat-earth, and often contains rootlets of the coal-forming plants.

Some seams have massive floors of sandstone, granite or igneous rocks, but usually the floors are composed of fireclay. The floors are unlaminated and consist of composite clots of materials breaking with an irregular or conchoidal fracture. Some of the irregularities are attributed to effects produced by the roots of the plants when living. The seat-earths are more or less plastic, and in deep mines rise in the roadways to give a curved floor.* On account of its softness, portions of the seat-earth may be added to the coal won.

Partings may consist of bind, shale or clunch (clay), sometimes more or less mixed with carbonaceous matter. Black shale, bat, bass, rattler are various names for carbonaceous shales, many of which resemble cannel coal in their appearance. Dirt partings may vary in thickness from a fraction of an inch to several feet, and the extent to which the dirt from partings can be excluded from the coal won depends on the thickness of the partings, and to some extent on their nature, *i.e.*, whether they are massive or friable. It also depends on the methods used to win the coal; with mechanical coal cutters the seam is often undercut in the seat-earth, or in a dirt parting, and the coal becomes more contaminated with free dirt.

In 1900, only about 1 per cent. of the coal mined in Great Britain was machine cut, in 1920 the total amounted to 13 per cent., but, by 1925, 20 per cent. of the output was won with the aid of machine cutting. Machine cutting has been applied chiefly to thin seams, particularly in Scotland, where the average thickness of the seams worked is only 3 ft. In Scotland, 50 per cent. of the output in 1925 was machine cut. In Northumberland, where the seams worked average 3 ft. 6 in. in thickness, 29 per cent. of the output was machine cut in 1925, but in South Yorkshire, with seams averaging 5 ft. in thickness, only 11 per cent. of the output was machine cut, and only 6 per cent. in South Wales and Monmouthshire with seams only slightly less thick, on the average, than in South Yorkshire. Hand-holing is usually impracticable in the floor of a seam, but machine-holing is readily accomplished. With thin seams, therefore, to obtain the maximum yield of lump coal, it is often the practice to undercut, by machines, in the seat-earths, and this practice inevitably leads to the inclusion of considerable quantities of dirt with the coal.

The analyses of a seat-earth and of a middle dirt parting have already been given (Table 2); the analyses of roof shale from the Top Hard seam, Notts., and from middle dirt bands of the King seam, Lancs., and the Wigan 4-ft. seam, Lancs.,† are recorded in Table 3, and compared with an analysis of the ash in the coal, uncontaminated with much free dirt.

* In some districts of deep pits which have been worked by the pillar and stall method, the seat-earth is forced up between the pillars making it impracticable to work the pillars left.

† Fuel Research Physical and Chemical Survey of the National Coal Resources, No. 6 and No. 10.

TABLE 3.—COMPARISON OF THE ANALYSES OF "FREE DIRT" AND "FIXED ASH" OF VARIOUS COAL SEAMS

	Top Hard Seam (Notts.).		King Seam (Lancs.).		Wigan 4-ft. Seam (Lancs.).	
	Shale.	Coal Ash.	Shale.	Coal Ash.	Shale.	Coal Ash.
Ash	93·8	9·2	91·2	Not determined	80·0	5·8
SiO_2	63·5	50·6	67·8		58·8	31·5
Al_2O_3	23·8	45·7	24·4		36·6	32·2
Fe_2O_3	4·6	0·9	2·7		1·8	21·0
CaO	1·5	1·4	0·8		0·9	3·3
MgO	1·0	0·3	0·8		0·2	6·2
Ratio SiO_2/Al_2O_3	2·67	1·11	2·77		1·61	0·98

It will be observed that the SiO_2/Al_2O_3 ratio for shales (free dirt) is usually high, but in the ash of coal uncontaminated by free dirt it is usually small.

The "free dirt" in coal from roof, floor and dirt partings has often a density of 2·3. When carbonaceous shales are included, the specific gravity is lower. In coal cleaning the chief aim is to remove the free dirt, but the heavier middlings present in raw coal, unless they can be crushed and rewashed, have often to be rejected with the dirt. The liability of a coal to contain middlings is indicated roughly by the ash content of the "pure coal" fraction floating in a liquid of S.G. 1·35. Coals fairly free from middlings (or with middlings of low ash content) yield material floating at S.G. 1·35 with only 1 or 2 per cent. of ash, but coals with a large middlings fraction usually have 5 per cent. of ash or more in the fraction floating at S.G. 1·35.

There are well-marked differences in the amounts and composition of the ash found in the four ingredients of banded bituminous coal, namely, vitrain, clarain, durain and fusain. The amount of ash in these ingredients usually increases in the order given, as is shown in Table 4.

The ash content of fusain, though usually high, may not affect the ash content of a coal sample appreciably, because fusain is usually only present in coal to the extent of a few per cent. Its great friability, however, leads to a concentration of this material in the fines or slurry, and in this connection it assumes a position of relative importance.

The ash contents of the coal at different horizons of a seam vary, as shown by the figures given in Table 5 (Sinnatt, *Journ.* Soc. Chem. Ind., : , 46, 244T). The horizons were separated at their natural partings formed by layers of fusain.

TABLE 4.—ASH CONTENTS OF THE INGREDIENTS OF BANDED BITUMINOUS COALS

Seam.	Vitrain per cent.	Clarain per cent.	Durain per cent.	Fusain per cent.
Parkgate, Yorkshire (a)	1·06	1·03	1·58	15·14
Parkgate, Yorkshire (b)	1·46	1·40	5·40	8·60
Parkgate, Yorkshire (c)	1·11	1·90	3·39	10·88
Top Hard, Notts.	0·90	1·30	7·80	13·80
Hamstead, Warwickshire	1·11	1·22	6·26	15·59
Cumberland (a)	1·19	1·30	1·94	0·77
Cumberland (b)	2·54	3·74	6·77	2·93
Furnace, Lancs.	1·20	2·59	8·66	6·25
Arley, Lancs.	0·87	1·15	2·56	4·72
Trencherbone, Lancs.	1·54	2·17	1·47	5·56
Wigan 5 ft., Lancs.	1·92	2·79	4·65	—
Wigan Yard, Lancs.	0·82	3·26	12·90	5·90

TABLE 5.—ASH CONTENTS OF DIFFERENT HORIZONS OF LANCASHIRE SEAMS

Seam.	Horizon.	Ash per cent.
Ravine.	Roof to 19 in.	6·0
	19–21 in.	31·4
	21–27 ,,	10·7
	27–29 ,,	38·3
	29–35 ,,	10·7
	35–49 ,,	7·2
	49–53 ,,	50·4
	53–61 ,,	6·7
	61–72 ,,	11·3
	Bottom dirt.	21·1
	Average for seam.	8·7
Wigan 5 ft.	Roof to 14 in.	7·4
	14–30 in.	3·0
	30–49 ,,	4·4
	Average for seam.	5·1

Distribution of Ash according to Sizes. If the methods of getting coal could enable a uniformly sized product to be loaded into the haulage tubs underground, the coal, as delivered to the consumer, would still be composed of differently sized lumps because of the crushing and fracturing necessarily involved in

any form of handling and transport. The largest lumps, those over, say, 6 in., would contain less ash than the smaller lumps because they would remain from those lumps which originally contained fewest cleavage planes, planes which represent or are caused by thin laminæ of mineral matter. In the smaller sizes the ash content is usually (though not universally) greater than in the lumps.

The distribution of ash according to size of coal may be illustrated by examples. The figures given by Drakeley (*Trans.* Inst. Min. Eng., 1919, 59, 71), are mean values for a number of Lancashire coals (Table 6).

TABLE 6.—DISTRIBUTION OF ASH IN LANCASHIRE COAL

Size of Coal.	Ash.
2½–2¼ in.	16·01 per cent.
2¼–2 ,,	16·42 ,,
2 –1¾ ,,	11·97 ,,
1¾–1½ ,,	15·93 ,,
1½–1¼ ,,	13·16 ,,
1¼–1 ,,	8·75 ,,
1 – ¾ ,,	10·03 ,,
¾– ½ ,,	15·43 ,,
½– ¼ ,,	16·89 ,,
¼–$\frac{1}{10}$,,	16·67 ,,
< $\frac{1}{10}$,,	20·35 ,,

The minimum amount of ash in these Lancashire coals is contained by coal less than 1¼ in. and greater than 1 in.

The ash analyses of the different sizes of a number of American (Pacific North-West) coals are given in Table 7. They show, in most cases, a progressive decrease in ash content from the largest sizes to the smallest studied.

TABLE 7.—ASH CONTENTS OF DIFFERENT SIZES OF COAL

Coal.	Rank.	3–1½ (in.).	1½–¾ (in.).	¾–⅜ (in.).	⅜–$\frac{3}{16}$ (in.).	$\frac{3}{16}$ in. to 20 mesh.	Below 20 mesh.
Beaver Hill	Sub-bituminous	37·2	31·6	28·3	27·5	—	—
Roslyn	Bituminous	—	36·8	21·7	17·5	16·6	16·8
Wilkeson, No. 3	Bituminous	47·6	38·6	30·0	25·0	—	—
Carbonado, No. 3	Semi-anthracite	29·2	25·7	24·6	22·9	24·4	27·6

The figures in Table 8 refer to a Vancouver coal. The bulk sample contained 28·1 per cent. of ash, and the coal was divided into ten portions by screening.

TABLE 8.—DISTRIBUTION OF ASH IN VANCOUVER COAL ACCORDING TO SIZE

Size of Coal.	Ash.
> $1\frac{1}{2}$ in.	16·6 per cent.
$1\frac{1}{2}$–1 „	18·2 „
1 – $\frac{3}{4}$ „	21·1 „
$\frac{3}{4}$– $\frac{1}{2}$ „	22·0 „
$\frac{1}{2}$– $\frac{3}{8}$ „	24·7 „
$\frac{3}{8}$– $\frac{1}{4}$ „	16·6 „
$\frac{1}{4}$– $\frac{1}{8}$ „	26·5 „
$\frac{1}{8}$–$\frac{1}{16}$ „	29·2 „
$\frac{1}{16}$–$\frac{1}{32}$ „	34·1 „
< $\frac{1}{32}$ „	38·4 „

The gradual increase in the ash content from size to size is very marked, the only exception being the fraction $\frac{3}{8}$ to $\frac{1}{4}$ in., which however, comprised only 0·92 per cent. of the whole.

A similar graduation is shown by an analysis of an American anthracite (Table 9).

TABLE 9.—DISTRIBUTION OF ASH IN AMERICAN ANTHRACITE ACCORDING TO SIZE

Size.	Ash.
$2\frac{1}{2}$–$1\frac{3}{4}$ in.	5·82 per cent.
$1\frac{3}{4}$–$1\frac{1}{4}$ „	10·30 „
$1\frac{1}{4}$– $\frac{3}{4}$ „	13·00 „
$\frac{3}{4}$– $\frac{1}{2}$ „	15·05 „
$\frac{1}{2}$– $\frac{1}{4}$ „	17·10 „

Generally, however, American anthracites from the Wyoming, Lehigh and Schuylkill fields have ash contents decreasing with size to about 28 mesh and increasing in the sizes less than this. An analysis of such an American anthracite is given in Table 10.

TABLE 10

Size of Coal.	Ash.
> $\frac{5}{16}$ in.	28·8 per cent.
$\frac{5}{16}$– $\frac{1}{4}$ „	16·5 „
$\frac{1}{4}$– $\frac{1}{8}$ „	17·0 „
$\frac{1}{8}$ in.–14 mesh	16·5 „
14–48 „	16·0 „
48–100 „	19·7 „
100–200 „	21·5 „
< 200 „	22·6 „

Where the coal of a seam is hard and the dirt associated with it relatively friable the finest sizes of coal contain the greatest amounts

of ash, but when the coal is itself friable, the increase in ash content with decreasing size may not be so marked, or the finer sizes may even contain less ash than the larger sizes. In Yorkshire, Derbyshire, Nottinghamshire, Lancashire, Staffordshire, Warwickshire, Cumberland and Scotland, the coals are usually hard, and a progressive increase in the ash content often occurs with decreasing size. This may be illustrated with the figures for Yorkshire coals in Table 11.

TABLE 11.—INCREASE IN ASH CONTENT OF COALS WITH DECREASE IN SIZE

Size (in.).	> 1–1	1–$\frac{3}{8}$	$\frac{3}{8}$–$\frac{1}{8}$	$\frac{1}{8}$–$\frac{1}{25}$	$\frac{1}{25}$	All Sizes.
	Ash per cent.					
High Hazel	1·8	5·6	9·0	15·7	20·8	7·2
Barnsley	—	9·9	12·9	16·3	20·8	12·1
Silkstone	6·4	8·1	14·4	21·5	11·8	14·3
Haigh Moor	—	8·8	17·4	22·3	21·8	15·3
Flockton	26·7	17·7	19·5	28·9	38·3	24·3
Seam C	19·8	25·0	23·7	29·0	39·3	26·3

In Durham the coals are often friable, and the increase in ash content with decrease in size may not be marked, as shown in the following examples (Table 12).

TABLE 12.—DISTRIBUTION OF ASH IN DIFFERENT SIZES OF DURHAM COALS

Size (in.).	Ash per cent. in Coal.			
	A	B	C	D
> 1	—	—	4·8	4·5
1 – $\frac{1}{2}$	—	—	18·6	5·3
$\frac{1}{2}$–$\frac{1}{4}$	13·0	—	19·5	5·6
$\frac{1}{4}$–$\frac{1}{8}$	12·8	11·2	15·6	6·9
$\frac{1}{8}$–$\frac{1}{16}$	12·8	12·9	15·8	4·5
$\frac{1}{16}$–$\frac{1}{64}$	14·6			
< $\frac{1}{64}$	15·9	13·3		

In South Wales and Monmouthshire, as well as in Germany, France and Belgium, the true coking coals are particularly friable,

and the decreasing sizes of coals usually show a decrease in ash content, as shown in the following examples (Table 13) :—

TABLE 13.—DISTRIBUTION OF ASH IN DIFFERENT SIZES OF WELSH AND BELGIAN COALS.

Size (in.).	2–1.	1–$\frac{1}{2}$.	$\frac{1}{2}$–$\frac{3}{8}$.	$\frac{3}{8}$–$\frac{1}{8}$.	$\frac{1}{8}$–$\frac{1}{16}$.	$\frac{1}{16}$–$\frac{1}{32}$.	< $\frac{1}{32}$.
S. Wales A .	18·9	15·3	15·0	10·6	8·5	7·2	6·8

Size (in.).	$2\frac{1}{4}$–$\frac{3}{4}$.	$\frac{3}{4}$–$\frac{3}{8}$.	$\frac{3}{8}$–$\frac{1}{50}$.	< $\frac{1}{50}$.
Belgium B . .	39·8	29·3	28·7	26·8
C . .	41·6	38·5	31·0	18·0
D . .	35·3	34·5	28·0	24·2

Sulphur.—The sulphur contained in coal is always an undesirable element, and often seriously impairs the value of the coal. All cleaning operations endeavour, as far as possible, to reduce the sulphur content of the cleaned product. In America, in some cases, the principal object of washing is to remove as much sulphur as possible from the coal, and the percentage recovery may be allowed to suffer to keep the sulphur content of the washed product to a low figure. The removable sulphur is usually present in coal as crystalline pyrites in lenticular inclusions on the bedding plane, or as thin sheets in the cleat. When the size of the raw material cannot be reduced to free the coal from the pyrites, the pyritic coal may be rejected with the dirt. On the other hand, a small body of pyrites buried in a relatively large lump of coal increases its specific gravity only slightly, and a high sulphur content may, in such a case, justify the expense of crushing before washing.

There are four principal forms in which sulphur occurs in coal, namely: (1) pyrites in lenticular masses and sheets; (2) finely-divided (microscopic) particles of pyrites disseminated throughout the coal material; (3) organic sulphur; (4) sulphate. Only the sulphur present as (1) can be removed by coal-cleaning processes.

Powell and Parr (*Bull.*, 111, Univ. of Ill. Eng. Expt. Stat., 1919) and Powell (Tech. Pap. No. 254, U.S. Bureau of Mines) record the following figures for the percentages of sulphur present in different forms in some American coals (Table 14).

TABLE 14.—FORMS OF SULPHUR IN AMERICAN COALS

Coal.		Sulphur per cent.			
		Total.	Pyritic.	Sulphate.	Organic.
S. Illinois	4	2·11	0·85	0·05	1·21
	5	1·23	0·31	0·25	0·67
	6	4·84	2·06	1·31	1·47
	7	3·12	1·36	0·31	0·55
Tennessee	1	0·85	0·29	0·01	0·55
	2	4·24	1·75	0·71	1·78
Pennsylvania	1	1·16	0·47	0·07	0·62
	2	1·68	0·79	0·23	0·66
W. Virginia		0·55	0·08	0·01	0·46
Kentucky		0·68	0·13	0·04	0·51
Kansas		3·02	1·99	0·32	0·71

Woolhouse (*Fuel*, 1925, 4, 456), using the Powell and Parr method for some British coals, found the percentages of sulphur present in different forms as follows (Table 15) :—

TABLE 15.—FORMS OF SULPHUR IN BRITISH COALS

Coal.	Sulphur per cent			
	Total.	Pyritic.	Sulphate.	Organic.
Anthracite (S. Wales)	1·06	0·75	0·08	0·23
Busty (Durham)	1·03	0·19	0·12	0·72
Silkstone (Yorkshire)	1·38	0·76	0·07	0·55
Parkgate X	2·00	1·04	0·07	0·89
Parkgate XI	2·01	1·09	0·09	0·83
Parkgate III	4·33	2·34	0·35	1·64
Parkgate VII	2·57	1·37	0·11	1·09
Parkgate IV	3·15	2·71	0·08	0·36
Derbyshire A	2·61	1·55	0·19	0·87
Derbyshire B	1·89	0·80	0·11	0·98
Thick (Staffs.)	4·30	2·11	0·32	1·87

The pyritic sulphur is decomposed on heating, one atom of sulphur being split from the FeS_2 molecule. The free sulphur atom combines with hydrogen to form hydrogen sulphide, in which form it is mostly expelled from the heated mass. Some of the

sulphur, however, combines with the lime present in the ash and is retained as calcium sulphide or calcium sulphate. During carbonisation an average amount of approximately 70 per cent. of the total sulphur is retained by the coke, some of which is removed as hydrogen sulphide during quenching.*

Foerster and Geisler (*Zeit. f. angew. Chem.*, 1922, 35, 193), working with German lignite, found that much of the organic sulphur was retained in the solid product of distillation as calcium sulphide.

Ditz and Wildmer (*Brenn. Chem.*, 1924, 5, 149, and 167) undertook a comprehensive study of the distribution of sulphur forms in an Arsa coal. The coal contained between 8 and 9 per cent. of sulphur, 96 per cent. of which was in the form of organic sulphur. Ditz and Wildmer found that, on carbonising the coal, much of the sulphur was retained in the coked product as sulphate. This they attribute to the presence of lime in the ash.

In Table 16 the removal of sulphur in washing a number of coals is recorded.

TABLE 16.—SULPHUR REMOVED BY WASHING

Coal.		Sulphur Content (per cent.).				
		Raw Coal.	Washed Coal.			Refuse.
			0–½ in.	½–1 in.	1–2 in.	
Lancashire (Drakeley)	1	0·50	0·47	0·42	0·41	0·89
	2	0·92	0·96	0·97	0·98	0·43
	3	4·73	1·82	2·27	2·81	10·89
	4	1·89	1·39	1·37	1·37	6·17
	5	1·67	1·40	1·37	1·36	6·32
	8	2·43	2·01	1·83	1·69	7·18
	12	1·82	1·33	1·16	1·13	6·09
American (Fraser and Yancey)	1	2·24	1·04			15·02
	2	2·59	0·90			8·48
	3	2·69	0·91			17·51
Pennsylvania		1·14	0·62			—
Kentucky		3·53	2·87			—
Tennessee		0·77	0·48			—
Washington		0·34	0·55			—
Montana		3·34	2·40			—
Ohio		1·05	0·89			—

* For the methods of determining the distribution of sulphur in coal, and for further information with regard to it, the reader should also consult: Powell, *Journ. Amer. Chem. Soc.*, 1923, 45, 1; Schellenberg, *Brenn. Chem.*, 1921, 2, 349 and 368; and the authorities already quoted.

Phosphorus.—Phosphorus occurs in coal in amounts varying from traces to 0·3 per cent. Durham coals usually contain less than 0·01 per cent.; many bright coals of South Yorkshire also contain less than 0·01 per cent. of phosphorus, but the hard coals usually have larger amounts. Phosphorus may often be a troublesome element in the use of metallurgical coke. This has been found to be the case for some Cumberland and South Wales coals, which are used for the production of coke for the manufacture of low-phosphorus pig iron used in acid-steel processes. For acid-steel manufacture a very low-phosphorus iron, and hence a low-phosphorus coke, is required. The manner of distribution of phosphorus in coal has not received the amount of study it deserves, and the methods of determining the total phosphorus, and of distinguishing between the organic and the inorganic phosphorus, are not altogether reliable. This is to some extent responsible for the fact that, at present, our knowledge of the removal of phosphorus in different washing processes is meagre, though it would appear that under the most favourable conditions the percentage removal of phosphorus is small. Cawley (*Fuel*, 1924, 3, 211) records the results of trials with a Baum washery, the analysis of the products being as follows:—

	Phosphorus per cent.
Nut coal	0·0306
Small coal	0·0263
Large shale	0·0183
Intermediate shale	0·0208
Small shale	0·0323
Slurry	0·0262

In this example the phosphorus appears to be concentrated in the large coal and the smaller dirt, though it is noteworthy that the largest size of shale contains the least of all the fractions. The results of further trials, using a froth flotation method, are summarised in Table 17.

TABLE 17.—DISTRIBUTION OF PHOSPHORUS IN FLOTATION PRODUCTS

	Original Coal.	First Froth.	Second Froth.	Third Froth.	Residue Coal and Dirt.	Dust from Washing Water.
Per c. of original coal (A)	—	11·7	8·45	53·33	17·95	2·80
Ash per cent.	7·78	2·76	2·98	4·72	39·54	38·3
Sulphur per cent.	1·79	1·51	1·36	1·57	1·86	2·2[illegible]
Phosphorus per cent.	0·0492	0·0342	0·0370	0·0515	0·0456	0·162
Per c. of original coal (B)	—	6[illegible]6	1·70	24·7	9·0	3·0
Ash per cent.	9·24	2·58	3·94	3·57	62·14	22·23
Phosphorus per cent.	0·0238	0·0200	0·0221	0·0224	0·0650	0·0485

These results suggest that a normal flotation process is no more successful than jig-washing for the removal of phosphorus from this coal. Better results were obtained with very fine coal, but the degree of fineness was such as to make the coal difficult to deal with on a practical scale. Cawley also gives figures which show that the phosphorus tends to be concentrated in the durain and fusain portions of some coals (Table 18).

TABLE 18.—DISTRIBUTION OF PHOSPHORUS IN THE BANDED INGREDIENTS OF BITUMINOUS COAL

	Phosphorus per cent.	Ash per cent.
Seam A.—Vitrain	0·0030	1·2
Clarain	0·0023	1·3
Durain	0·0024	1·9
Fusain	0·0056	0·8
Seam B.—Vitrain	0·0648	2·5
Clarain	0·0802	3·7
Durain	0·2000	6·8
Fusain	0·2200	2·9

The concentration of phosphorus may vary in different benches of the seam, this being illustrated by figures for the Pittsburgh seam, Pennsylvania, recorded by Fulton (*Coke*, Scranton, Pa., 1905, p. 41). In the Connellsville region, the seam is divided into two portions, each of which has three subdivisions or benches which contain the following percentages of phosphorus :—

	Per cent.
In the Upper Portion—	
Top, 1 ft.	0·062
Middle, —	0·036
Bottom, 1 ft.	0·028
In the Lower Portion—	
Top, —	0·039
Middle, 2 ft.	0·019
Bottom, 3 ft.	0·009

It would appear therefore that selective mining of this seam, if practicable, would reduce the phosphorus content of the coal.

Davies (*Fuel*, 1926, 5, 550) examined the distribution of phosphorus in a number of Welsh coals. He states that the limiting amount of phosphorus allowed in coke in South Wales is 0·03 per cent. Many South Wales coals contain relatively large amounts of phosphorus and can only be used by blending with low-phosphorus coals. In past years coking coal mixtures of sufficiently low-phos-

phorus content were obtained by mixing high-phosphorus coals with coal from No. 2 and No. 3 Rhondda seams, with phosphorus contents less than 0·010 per cent. The depletion of the reserves of these two seams has accentuated the difficulties experienced, and considerable expense has been incurred in purchasing suitable coals of low-phosphorus content.

Davies divided a number of coals into different sizes, but did not find any regular variation of phosphorus content in the different grades. By separating the fractions in liquids of different density, however, a progressive increase was found in the phosphorus contents of coals floating in liquids of higher density. Some of his results are recorded in Table 19.

TABLE 19.—PHOSPHORUS CONTENTS OF SOUTH WALES COALS

Seam.	Floats at S.G.									Sinks at S.G. 2·4.
	1·3.	1·325.	1·35.	1·45.	1·60.	1·8.	2·0.	2·2.	2·4.	
C.P.W.	0·015	0·043	0·100	0·147	0·410	1·163	0·267	0·092	0·042	0·111
C.T.N.W.	0·004	0·009	0·039	0·097	0·298	0·658	2·452	1·114	0·316	0·056
C.N.E.	0·005	0·009	0·038	0·097	0·150	0·250	0·334	0·120	0·092	0·164
C.D.S.W.	0·009	0·012	0·014	0·017	0·028	0·117	0·117	0·089	0·056	0·139

The fractions of S.G. 1·6 to 2·0 contained the largest amounts of phosphorus. Davies calculated that, to produce coal containing 0·025 per cent. of phosphorus, the percentage recovery for different coals and their ash contents would be :—

	G.P.	G.L.F.	C.P.W.	C.T.N.W.	C.N.E.
Yield per cent.	48·0	90·0	67·5	80·0	78·0
Washed Coal Ash per cent.	2·75	3·75	2·00	1·70	1·70
Refuse, Ash per cent.	21·6	31·4	28·0	52·2	56·5

To produce washed coals of such low ash contents (although the phosphorus contents would be satisfactory) was considered to be justifiable only when it was possible to dispose of a separate middlings fraction.

Salt.—Salt (sodium chloride) is a troublesome constituent of many coals, particularly in South Yorkshire, North Staffordshire, and in certain districts of Germany. It has been stated by Fearnsides that coals mined at a depth below 1,000 ft. are usually salty. The salt accumulated in the coal-forming material which was deposited in brackish water. The bright portions of coal (clarain and vitrain) usually contain less salt than the dull portion (durain).

Bradley (*Fuel*, 1928, 7, 31) records figures for the distribution of salt in different sizes of a South Yorkshire coking slack before and after washing. His figures are reproduced in Table 20.

TABLE 20.—DISTRIBUTION OF SALT IN DIFFERENT SIZES OF RAW AND WASHED COAL

Size.	Raw Coal. Salt per cent.	Washed Coal. Salt per cent.	Per cent. of Salt removed.
> $\frac{3}{8}$ in.	0·29	0·20	31
$\frac{3}{8}$–$\frac{1}{4}$ in.	0·30	0·24	20
$\frac{1}{4}$–$\frac{1}{8}$,,	0·30	0·20	33
$\frac{1}{8}$–$\frac{1}{16}$,,	0·39	0·20	49
$\frac{1}{16}$ in.–30 mesh	0·54	0·21	61
30–90 mesh	0·71	0·26	63
< 90 mesh	1·10	0·58	48

In the raw coal the salt content was almost uniform for sizes above $\frac{1}{8}$ in., but rapidly increased for sizes less than $\frac{1}{8}$ in., until it was over 1 per cent. in the smallest size. Possibly the porous ingredient of coal, fusain, carries the greatest percentage of salt (*cf.* Louis, *Chem. and Ind.*, 1927, 46, 547). The salt content of the washed coal was approximately constant for all the sizes except the smallest, and, therefore, more salt was removed from the finer sizes, which expose a greater percentage surface to the solvent action of the washery water. Bradley also gave figures for the distribution of salt in five samples of unwashed coking slack (South Yorkshire) which are reproduced in Table 21.

TABLE 21.—DISTRIBUTION OF SALT PER CENT. IN DIFFERENT SIZES OF COAL

Size.	E.	B.	A.	C.	D.
> $\frac{3}{8}$ in.	0·08	0·15	0·21	0·29	0·35
$\frac{3}{8}$–$\frac{1}{4}$ in.	0·15	0·23	0·25	0·30	0·34
$\frac{1}{4}$–$\frac{1}{8}$,,	0·15	0·18	0·30	0·31	0·40
$\frac{1}{8}$–$\frac{1}{16}$ in.	0·16	0·19	0·30	0·34	0·42
$\frac{1}{16}$ in.–30 mesh	0·16	0·19	0·36	0·41	0·47
30–90 mesh	0·19	0·21	0·48	0·62	0·64
< 90 mesh	0·13	0·27	0·79	1·04	1·08
Total	0·14	0·18	0·36	0·37	0·45

The coals are arranged in order of increasing salt content of the total samples, and this order is maintained for every size of coal. The salt is therefore distributed throughout the coal substance. The distribution is again fairly uniform for sizes above ⅛ in., or in some cases above $\frac{1}{16}$ in., but below these sizes (and particularly below 30 mesh size) the increase in salt content is very marked, which supports the view that the salt is more concentrated in the fusain fraction (which accumulates in the finest sizes) than in the rest of the coal.

It has been found that when coals containing more than about 0·05 per cent. of sodium chloride are coked in coke ovens, the refractory linings suffer from corrosion, which may be so severe as to necessitate relining after a few months of use. The corrosion reveals itself when pieces of the brick of the thickness of perhaps ½ in. peel off the walls, leaving a roughened surface which may be further attacked, and which in itself increases the difficulty of pushing the coke from the ovens.

The chlorides in the coal are not entirely present in a soluble form. J. W. Cobb (*Gas World*, 1916, Coking Section, April) showed that with one coal 0·38 per cent. of salt was extractable with water, but a further 0·41 per cent. was removed in dilute nitric acid; with another coal the amounts were 0·36 and 0·26 per cent. respectively. During the short time the coal is in contact with the water in a washer, and because of the relatively small surface that the coal offers, the removal even of soluble salt from coal is imperfect. By recirculation of the washing water the concentration of salt in the water increases.

In Table 22 the percentage amounts of salt in washed and unwashed coking slacks are recorded :—

TABLE 22.—PER CENT. OF SALT IN COALS BEFORE AND AFTER WASHING

Plant.	In Coal entering Washer.	In Slack leaving Washer.	In Slack after Drainage.	In Washing Water.
S. Yorkshire 1	0·217	0·107	0·06	0·7–1·0
S. Yorkshire 2	0·37	—	0·18	0·15
S. Yorkshire 3	0·39	—	0·19	0·15–1·0
S. Yorkshire 4	0·60	—	0·15	—
W. Yorkshire	0·46	—	0·28	—
Derbyshire 1	0·488	—	0·378	0·246

The Effect of Washing on the Fusibility of Coal Ash.—The fusibility of their ashes is one of the chief objections to many coals,

A general relationship exists between the softening temperature of the ash of a coal and its liability to clinker. Not only does clinker clog the firebars of a grate and necessitate the use of an excessive draught, but it frequently results in a direct loss of fuel by enclosing particles of unconsumed combustible matter in a molten or semi-molten mass of inorganic matter. For pulverised-fuel firing perhaps the commonest fault of some coals is the fact that their ashes have low fusing points, with the result that trouble is experienced in maintaining the refractory lining of furnaces, and the material in metallurgical furnaces suffers from ash contamination and scaling. In a boiler, the efficiency of heat transfer to the water tubes is decreased by the presence on them of a coating of semi-molten ash of low thermal conductivity.

The effect of cleaning a coal on the melting-point of its ash is therefore an important consideration. Sinnatt and Wood (*Trans.* Inst. Min. Eng., 1923, 3–4, 66, 58) suggest that, on occasion "excessive purification may produce coal yielding ash, the melting-point of which is relatively low, and thus reduce the value of the purified coal for certain industrial purposes."

Sinnatt, Owles and Simpkin (*Journ.* Soc. Chem. Ind., 1923, 42, 266T) have shown that, after screening and washing, the melting-points of the ashes produced from different sizes of the products may vary. Their results for three coals are quoted in Table 23.

TABLE 23.—MELTING-POINTS OF ASH (DEGREE CENT.)

Grade of Coal.	Seam A.	Seam B.	Seam C.
Lump . .	1,345	1,280	1,345
Slack . .	1,240	1,240	1,300
Slurry . .	1,190	1,200	1,180

In general, the melting-point of a coal ash is relatively low if it contains comparatively large quantities of iron, lime or magnesia,

TABLE 24.—MELTING-POINTS OF ASHES OF VITRAIN, CLARAIN, DURAIN AND FUSAIN (DEGREE CENT.)

Vitrain.	Clarain.	Durain.	Fusain.
1,340	1,310	1,430	1,220
—	1,260	1,430	1,200
1,310	1,320	1,450	1,200

whereas with high contents of silica and alumina, an ash has a relatively high melting-point.

Bright coals (clarain and vitrain) usually contain large quantities of iron which is, however, almost absent from the ash of dull, hard coal (durain). The melting-points of durain ashes is, therefore, usually higher than that of clarain and vitrain ashes, as shown in Table 24.

Saline deposits from percolating waters may result in fusain having a high content of iron, lime, magnesia and alkalies, which reduce the melting-point of the fusain ash. Fusain ash has the lowest melting-point, and one precaution which therefore suggests itself in preparing a coal of which a high ash melting-point is desirable, is the removal of the finest grades of coal, which usually contain the bulk of the fusain.

CHAPTER II

THE EXAMINATION OF COAL PRIOR TO CLEANING

GENERAL

THE removal of dirt from the coal won in the pit begins at the coal face, when some of the lumps of rock or shale mixed with the coal are discarded. In deciding to clean the coal by hand-picking or by automatic means, a colliery company must first consider the nature of the seams available, and must decide what proportion of the dirt can be removed underground, and to what extent it may be advisable to win the whole of the seam, and to remove the impurities from it above ground. It may occur that a thin band of coal is separated from the main seam by a thick band of shale, and, in these circumstances, it may be more profitable to leave the thin coal band rather than to cut the dirt band and endeavour to separate the impurities from the coal, either underground or at the surface. The consideration of this point involves solely questions of market values and mining difficulties, and the quality of the coal in the thin band and the character of the dirt and of the roof are of primary importance.

Decisions on these points being reached, the next point to consider is the degree to which cleaning can be effected by hand-picking, and how much of the small coal must be cleaned by automatic or mechanical means. In some cases it is desirable to wash all the coal below 4 in., in others hand-picking provides a sufficient degree of cleaning down to a size of 2 in., and in other cases it may only be considered necessary to wash the slack below, say, $\frac{5}{8}$ in. These matters are essentially ones of cost and of market values. Hand-picking is usually inefficient on sizes below 4 in., and, though the raw coal below 4 in. may contain only 7 or 8 per cent. of ash, the cost of washing all the coal below 4 in. is seldom other than a profitable investment on account of the greater utility of the cleaner coal. In any event, it is usually desirable to wash the slack coal, for the smaller sizes are frequently the highest in ash content. If all the coal below 4 in. contains 8 per cent. of ash, it may be possible to reduce the average ash content by 2 or 3 per cent. by washing only the smaller sizes, below, say, $\frac{5}{8}$ in., the nuts (4 to $\frac{5}{8}$ in.) being by-passed. This will depend upon the average distribution of the coal according to size, and especially upon the relative proportions of " pure " coal, " pure " dirt and middlings in the nuts and in the slack.

In order to decide what portions of the raw coal to pass through the washer, the coal too small to be hand-picked must be examined in two ways. Firstly, an average screening analysis of the raw coal over a period must be made, and, secondly, it is necessary to know the fractions into which each separate size of coal can be divided by washing. For the latter purpose several methods of examination are available, namely :—

1. Washery trials,
2. Float and sink analysis,
3. The Henry tube method.

WASHERY TRIALS

One method of determining the maximum degree of cleaning to which a given coal sample can be subjected is to wash the coal several times, or for prolonged periods, and assume that the product is "pure" coal (purity being measured by the comparative absence of "free dirt"). In a jigging plant the washing operation can be carried on for a long time and a sample of the top layer in the washing box may be regarded as coal in the purest state to which that washer can treat it.

This method is a fairly reliable one and has the advantage that it is a trial under working conditions, though perhaps more severe than the normal. Its drawback is that it makes no allowance for daily fluctuations in the feed coal, and gives no indication of the nature of alternative products that might be produced to meet market requirements.

Although usually the examination of coal before washing does not require a large-scale trial, there are many cases in which such a trial is desirable to indicate the amount of disintegration of the shale and fracture of the coal sustained in practical operation. This information is often required in calculating the necessary provision for water clarification and for the settling-out of slurry.

FLOAT AND SINK METHODS

A more satisfactory means of determining the washing characteristics of a coal is provided by float and sink trials. The majority of cleaning processes depend for their action upon the difference in density of coal and its impurities, the lighter coal being recovered, whilst the heavier mineral matter is discarded. It has also been found that float and sink methods may serve as a useful guide for the control of flotation processes, despite the fact that such processes do not depend on density differences for their working.

To determine the washability of a given representative sample of a raw coal, the coal is graded according to size. Each portion is then tested in liquids of different density, the amounts of "float" and "sink" being determined. Sizing is desirable, for it facilitates

the separation of the coal and the impurity, and at the same time provides information with regard to the concentration of impurities in different sizes of the coal. In any event, before testing, the coal should be freed from dust.

If the coal is found to contain large quantities of middlings, it may be desirable to crush them and test the crushed material separately. This will indicate whether crushing and rewashing the middlings is likely to be profitable.

Sizes for Testing.—Suitable sizes for grading the coal before float and sink testing are :—

Over 1 in.
1 to $\frac{3}{8}$ in.
$\frac{3}{8}$ to $\frac{1}{8}$ in.
$\frac{1}{8}$ in. to 1 mm.
Below 1 mm.

When the nuts and fines are washed separately, the sizes should include that at which they will be screened before washing, and suitable sizes are: over 1 in., 1 to $\frac{5}{8}$ in., $\frac{5}{8}$ to $\frac{3}{8}$ in., $\frac{3}{8}$ to $\frac{1}{8}$ in., $\frac{1}{8}$ in. to 1 mm., below 1 mm. Similarly, when all sizes of the raw coal are treated in one washer and the fines are screened out from the washed coal and rewashed, the screen size selected should also be included. In some washers, close sizing of the feed is required, each size being fed to a separate unit. In these circumstances, closer sizing ratios may be desirable.

Liquids to Give Required Specific Gravities.—The most suitable liquids for use as separating medium are carbon tetrachloride or solutions of calcium or zinc chloride in water. Carbon tetrachloride has a specific gravity of 1·61 at 15° C. To obtain solutions of different densities, the carbon tetrachloride is diluted with toluene (specific gravity 0·872 at 15° C.). Solutions of given density (at 15° C.) are given by mixtures of the following composition :—

Specific Gravity.	Vol. of Carbon Tetrachloride per cent.	Vol. of Toluene per cent.
1·20	44·4	55·6
1·25	51·2	48·8
1·30	58·0	42·0
1·35	64·8	35·2
1·40	71·5	28·5
1·45	78·1	21·9
1·50	85·1	14·9
1·55	91·9	8·1
1·60	98·7	1·3

Benzene is also a convenient diluent for the carbon tetrachloride. When an aqueous solution of salt is used as the separating medium, a concentrated solution is diluted with water to the required density. The densities of aqueous solutions containing different percentages of anhydrous zinc or calcium chloride are as follows :—

S.G. at 15° C.	Gm. Zinc Chloride per 100 gm. of Solution.	Gm. Calcium Chloride per 100 gm. of Solution.
1·25	26	26
1·30	31	31
1·35	35	35
1·40	39	40
1·45	42	—
1·50	46	—
1·55	49	—
1·60	52	—
1·65	55	—
1·70	58	—
1·74	60	—

For bituminous coals, sufficient information will usually be obtained by making the density of the final testing liquid 1·60, but frequently specific gravities up to 1·75 or 1·80 are employed. For general use, specific gravities of 1·35, 1·40, 1·50 and 1·60 are satisfactory. If the coal contains more than 10 to 12 per cent. of specific gravity 1·40 to 1·60, a further test should be done at 1·80. With some anthracites it may be desirable to use still higher specific gravities. For specific gravities above 1·60, methyl iodide (S.G. 2·285 at 15° C.), methyl bromide (S.G. 1·69 at 15° C.) or bromoform (S.G. 2·90 at 15° C.) are preferable, bromoform being the cheapest. With these liquids, chloroform or benzene may be used as diluent to obtain intermediate specific gravities. Concentrated sulphuric acid is occasionally used for specific gravities between 1·70 and 1·84, but it cannot be recommended for ordinary use, for, in addition to the difficulties of manipulation, the dirt frequently disintegrates rapidly and the liquid becomes warm (altering its specific gravity) unless the particles are well dried.

Organic liquids are always preferable to aqueous solutions, for the fractions need not be washed to remove traces of the solution, and drying is accomplished quickly on exposure to the air. Testing is completed much more rapidly, and the expense is little greater.

Quantity of Coal to Use.—If the coal is not divided into a number of sizes, the particles greater than about ½ in. should be removed

and tested separately. At least 600 gm. of material less than ⅛ in. should be used to ensure reliable results.

Blyth and O'Shea (*Trans.*, Inst. Min. Eng., 1919, 57, 261), suggest a rational basis for the amount taken and recommend that enough of each size should be tested to ensure that at least 2,000 particles are present in each sample. The approximate quantities recommended for use with different sizes of coal are as follows :—

Size (in.).	Weight.
Under ⅛	30 gm.
⅛ to ⅜	400 ,,
⅜ ,, ⅝	2,000 ,,
⅝ ,, 1	32 lb.
1 ,, 2	2 cwt.
Over 2	4 ,,

To simplify calculation, exact weights (*e.g.*, 400 gm.) are sometimes taken. But this is a mistake. The sample is usually obtained by quartering a larger bulk, and the whole of two opposite quarters should be used. The use of an exact weight neutralises many of the advantages of careful sampling.

Method of Conducting Tests.—The sample is placed in the liquid with the lowest specific gravity (usually 1·35) agitated and allowed to settle. The fraction which floats is collected, washed, if necessary, dried and its air-dry weight determined. The sinkings are placed in the liquid of next higher specific gravity (1·40), separated as before, and the floatings again collected. The sinkings are treated in a solution of 1·50 specific gravity, and the operation is continued until the last separation is effected by the liquid of highest specific gravity. The ash content of each fraction should then be determined.

The sample for testing should be roughly air-dried, so that the free moisture is removed. Otherwise the specific gravity of an aqueous solution may vary after being used several times, and, if organic liquids are used, the water complicates the separation by introducing surface tensional effects, and it also dirties the liquid.

Coals which are fairly moist when mined (some American coals, for example) should, however, be tested in the same condition as when mined, and they may be saturated with water by boiling or by standing in water overnight. An aqueous solution of calcium or zinc chloride should then be employed, and its specific gravity should be checked frequently.

When aqueous solutions are used, it is preferable to agitate small coal with the liquid for a definite period of time, and the time allowed for settling should also be fixed. Five minutes' agitation

and two minutes' settling is satisfactory for most coals, but shorter periods are usually sufficient. Some coals slowly absorb water, and their specific gravities are thereby increased. Consequently, if too long a time is allowed, some of the floatable material may be converted into sinkings. If, on the other hand, too short a time of agitation is allowed, the smallest particles may not be completely freed from bubbles of air, and material that should sink may be floated.

Various forms of apparatus have been suggested for float and sink tests, but a beaker and spoon are all that are usually required. Particles above 1 in. may be tested three or four at a time, the floats being removed and the sinks being allowed to collect at the bottom of the vessel and removed periodically. Particles under 1 in. may be tested a handful at a time. Particles less than ¼ in. in size should be agitated for the stated period, the floats from one handful being removed before the next handful is added.

With small particles, the layers of floats and sinks usually retain some particles which should properly pass into the other layer, but which remain mechanically entangled with the floats or with the sinks. When the layers have separated, therefore, the floats should be stirred with a spoon or rod to allow heavy particles entrapped to be released. The floating layer may then be removed with a spoon. When the layer has been removed almost completely, the sinking particles should be stirred to allow light particles to escape. The remaining floats may then be removed, and the sink may be collected by filtering.

Coal has two specific gravities, one a "true" and the other an "apparent" specific gravity. The true specific gravity is that of

TABLE 25.—EFFECT OF AIR-DRYING ON APPARENT SPECIFIC GRAVITY OF SOME AMERICAN COALS

Coal.	Coal as Mined.		Air-dried Coal.		Authority.
	S.G.	Per cent. Moisture.	S.G.	Per cent. Moisture.	
A . . .	1·30	13·9	1·23	6·7	McMillan and Bird*
B . . .	1·30	16·5	1·23	10·3	,,
C . . .	1·32	22·3	1·29	10·0	,,
D . . .	1·29	29·7	1·23	9·8	,,
E . . .	1·35	4·5	1·33	0·9	,,
F . . .	1·30	—	1·12	—	Nebel†
G . . .	1·30	—	1·20	—	,,

* *Bull.* 28, University of Washington Engineering Expt. Station.
† *Bull.* 89, University of Illinois.

the coal substance freed from air and moisture and is of little value for a study of coal-cleaning; the apparent specific gravity includes the air and moisture in the coal material, and is the one operative in float and sink testing. The apparent specific gravity may be calculated from the formula :—

$$S = \frac{Wa}{Wa - Ww},$$

in which Wa = its weight in air and Ww = its weight in water.

The apparent specific gravity of a number of American coals before air-drying and after air-drying is recorded in Table 25.

It may be seen that the elimination of moisture does not decrease the specific gravity at all proportionately, but does invariably cause some decrease.

It may be noted in passing that the effect of air drying on the apparent specific gravity of coals is of more importance with American coals than with British coals, because American coals usually have a much higher moisture content. Thus in South Wales, Yorkshire and Durham, from which coalfields over half of the yearly British coal output is mined, it is rare to find a coal containing more than 3 per cent. of moisture, and for coal from these fields the effect of air-drying on the apparent specific gravity would consequently be almost negligible.

General Considerations.—The approximate specific gravities of the materials found in raw coal are given in Table 26.

TABLE 26.—SPECIFIC GRAVITIES OF MATERIALS OCCURRING IN RAW COAL

Substance.	Specific Gravity.
Bituminous coal	1·12 to 1·35
Middlings	1·35 ,, 1·6
Carbonaceous shale	1·6 ,, 2·2
Shale	2·0 ,, 2·6
Clay	1·8 ,, 2·2
Pyrites	4·8 ,, 5·2
Sandstone	2·2 ,, 2·6
Quartz	2·5 ,, 2·7
Mica	2·7 ,, 2·[illegible]
Ankerites	2·5 ,, 2·7

"Middlings" are particles which consist partly of coal and partly of dirt. Usually the coal and dirt occur in layers, and may be separated mechanically. Their specific gravity varies according to the proportions of coal and dirt, but it is seldom profitable intentionally to recover with the clean coal any of the middlings with a higher specific gravity than 1·60. In America, the

term "bone" coal is used to designate a dull hard portion of the coal which has a much higher ash content and a higher specific gravity than the adjacent bright coal. Thus the "bone" coal of the Upper Freeport bed, Pennsylvania, has a specific gravity varying from 1·40 to 1·60, whilst the ash content is, on the average, about 30 per cent. (Yancey, *Bull.* 16, Carnegie Inst. of Tech., 1924). Bone coal of Pacific North West coals, varies in specific gravity from 1·35 to 1·70, with an ash content of from about 18 to about 40 per cent. (MacMillan and Bird, *loc. cit.*).

The dull hard portion—durain—of British bituminous coals does not contain as much ash as American "bone" coals. In Table 27 the ash contents and specific gravities of a few British durains are recorded.

TABLE 27.—ASH CONTENTS AND SPECIFIC GRAVITIES OF BRITISH DURAINS

Seam.	Ash per cent.	Specific Gravity.	Authority.
Hamstead (Warwick.)	3·6	1·39	Tideswell and Wheeler
Top Hard (Notts.)	7·8	1·47	Baranov and Francis
Parkgate—A (S. Yorks.)	2·0	1·28–1·33	—
,, B (,,)	3·6	1·31–1·33	—
Eckington (Derby.)	2·1	1·31	Greenwood
Arley (Lancs.)	4·5	—	Simpkin
Wigan 4 ft. (Lancs.)	1·3	—	Simpkin
Wigan 5 ft. (Lancs.)	11·8	—	Simpkin

The apparent specific gravity of a coal varies with the degree of humidity of the coal and to some extent with the size of the coal; the larger the coal, the lower, in general, is its apparent specific gravity. A lump of coal may, on the other hand, enclose a body of mineral matter, and although its specific gravity falls well within the limits for coal, it may actually be a very impure sample. Thus a block of coal of specific gravity 1·35 may consist of 88 per cent. of coal of specific gravity 1·25 and 12 per cent. of shale of specific gravity 2·15.

The Use of Float and Sink Tests.—In coal-cleaning practice, float and sink tests are utilised for the examination of the washability of a coal and for the control of coal-washing plants. Their utilisation for control testing is described at length in Chapter XXX, and the present description may, therefore, be confined to their use

in the preliminary examination of the coal to determine its washing characteristics.

In the preliminary examination, float and sink tests serve three purposes, namely :—

1. To suggest the type of cleaning process most suitable for the coal.

2. To decide the nature of the products that can most suitably be prepared from the coal.

3. To interpret the results of practical washing tests, and to indicate the type of washer which will give the most profitable results.

The third purpose, to indicate the most suitable type of washer, is related to the first, to suggest the best type of process, but the two are not identical. Float and sink tests may, for example, suggest that a process dependent solely on density differences, is better for a certain coal than a froth-flotation process, thus fulfilling the first purpose. Subsequently, float and sink tests may be applied to the products obtained in two different "gravity" processes, and thus indicate which of the two enables the greater recovery of clean coal to be accomplished.

1. *The Choice of Washing Process**.—The division of the coal into fractions, each of which contains particles similar in many respects, enables visual examination of the coal to be made more easily. Thus the presence of flat shale particles, which are difficult to remove in cleaning by certain processes, may be detected, and the middlings fraction may be examined for the presence of intergrown particles from which coal could be liberated by crushing.

To demonstrate the use of float and sink tests in the examination for coal, it is convenient to give the results of examination of certain coals and to indicate the conclusions drawn therefrom.

The results of float and sink tests on a Yorkshire slack coal are given in Table 28, the dust below 1 mm. in size being excluded.

TABLE 28—FLOAT AND SINK RESULTS, YORKSHIRE COAL

Weights Expressed as Per Cent. of Total Sample

Size.	Per cent. of Total.	Per cent. Ash in Fraction.	S.G. <1·35.		S.G. 1·35–1·4.		S.G. 1·4–1·5.		S.G. 1·5–1·6.		S.G. >1·6.	
			Wt. per cent.	Per cent. Ash.	Wt. per cent.	Per cent. Ash.	Wt. per cent.	Per cent. Ash.	Wt. per cent.	Per cent. Ash.	Wt. per cent.	Per cent. Ash.
> 3/8 in.	47·0	8·8	41·2	1·5	0·6	12·6	Nil.	—	Nil.	—	5·2	66·4
3/8–1/8 in.	30·6	17·4	24·2	1·8	0·3	10·0	0·3	16·4	Nil.	—	5·9	82·0
1/8 in.–1 mm.	22·3	22·2	15·6	1·4	0·5	10·0	0·2	15·7	0·1	20·1	5·9	79·0
Total	100·0	14·4	81·0	1·6	1·4	10·7	0·5	16·1	0·1	20·1	17·0	76·2

* For a full comparison between coal-washing processes, see Chapman and Wheeler, J. Soc. Chem. Ind., 1927, 46, 229T.

From these results it is evident that only 2·0 per cent. of the coal has a specific gravity between 1·35 and 1·6. The sinks at 1·6 contain 76·2 per cent. of ash, and therefore consist of a heavy material, probably almost a pure shale, which could easily be separated from the light coal and the middlings. The coal is therefore an easy one to wash, and a simple washer, with a high capacity and small operating space (*e.g.*, Baum) would be suitable.

In Table 29 the results for a Westphalian coal are given.

TABLE 29.—FLOAT AND SINK RESULTS, WESTPHALIAN COAL

S.G.	Per cent of Total Sample.	Ash per cent. in Fraction.	Cumulative.	
			Wt. per cent.	Ash per cent.
< 1·3	43·5	2·0	43·5	2·0
1·3–1·4	9·5	9·8	53·0	3·4
1·4–1·5	14·5	20·0	67·5	6·9
1·5–1·6	10·0	31·8	77·5	10·1
1·6–1·7	5·7	40·5	83·2	12·2
1·7–2·2	8·8	54·0	92·0	16·2
> 2·2	8·0	65·5	100·0	20·1

This is obviously a difficult coal to wash. Nearly 25 per cent. of it has a specific gravity between 1·4 and 1·6, and there is a gradual transition from coal, with the lowest specific gravity, to middlings and to dirt, with a specific gravity greater than 1·6 or 1·7. In order to wash it successfully a process must be used which is able to give a sharp differentiation between particles differing little in specific gravity and, preferably, one capable of yielding a variety of products. In operation the middlings fraction would be crushed and rewashed, and the process must therefore be able to deal efficiently with small particles. Probably the Rheolaveur would be the most suitable.

Float and sink results indicate the extent of the benefit to be derived from crushing the raw coal or by collecting a separate middlings fraction, crushing it and rewashing. For this purpose float and sink results are performed on the raw material and on the crushed material. These results discover the amount of " coal " and " shale " freed from interstratified particles by crushing.

Draper (*Proc.*, S. Wales Inst. of Eng., 1919, 35, 21) records a case in which lump coal containing 16·4 per cent. of ash required to be crushed to pass through a ⅛ in. screen before any improvement

The coking coal produced, containing 7·86 per cent. of ash and 1·03 per cent. of sulphur, may be a satisfactory quality of coking coal, but the yield is low, nearly 20 per cent. of the raw coal being rejected. To overcome this loss, the third method (*c*) of preparing the coal may be employed. Instead of aiming exclusively at a coking coal, two products may be obtained—one, a good quality coking coal; and the other a coal suitable for use as a steam coal or a coal for other purposes.

If, for example, that portion of fraction A which floats on a liquid of 1·34 S.G. is recovered as a coking coal, the refuse obtained, that portion sinking in a liquid of 1·34 S.G., may be rewashed and the 1·425 float be recovered as steam or domestic coal. Fraction B may then be added either to the coking coal or to the washed refuse. In these circumstances the possible operating results are shown in Tables 36 and 37. In Table 36 fraction B is added to the coking coal; in Table 37 it is added to the steam coal.

TABLE 36.—WASHING CHART FOR COKING COAL AND STEAM COAL. METHOD (*c*)

	Weight per cent. of Total Sample.	Ash per cent.	Sulphur per cent.
Coking Coal—			
Fraction A, floating at 1·34	48·66	5·50	0·97
Fraction B, raw coal (− $\frac{1}{18}$ in.)	18·79	8·10	1·28
Resultant coking coal	67·45	6·23	1·06
Steam coal—			
Fraction A, float at 1·425, sink at 1·34	25·14	15·80	1·03
Refuse	7·41	42·75	4·16
Original coal	100·00	11·34	1·28
Total yield	92·6 per cent.		

In both these cases the amount of coal recovered is 92·6 per cent. of the raw coal. In the first case the coking coal amounts to 67·45 per cent., containing 6·23 per cent. of ash and 1·06 per cent. of sulphur, and the steam coal amounts to 25·14 per cent., containing 15·8 per cent. of ash and 1·03 per cent. of sulphur.

In the second case the coking coal amounts to 48·66 per cent., but the quality is improved, the ash content being 5·50 per cent. and the sulphur content 0·975 per cent. The yield of steam coal is increased to 43·93 per cent. and its ash and sulphur contents are 12·25 and 1·11 respectively.

TABLE 37.—WASHING CHART OF COKING COAL AND STEAM COAL. METHOD (*c*)

	Weight per cent. of Total Sample.	Ash per cent.	Sulphur per cent.
Coking coal—			
Fraction A, floating at 1·34	48·66	5·50	0·97
Steam coal—			
Fraction A, float at 1·425, sink at 1·34	25·14	15·80	1·03
Fraction B, raw coal ($-\frac{3}{16}$ in.)	18·79	8·10	1·28
Resultant steam coal	43·93	12·25	1·11
Refuse	7·41	42·75	4·16
Original coal	100·00	11·34	1·28
Total yield	92·6 per cent.		

The fourth possible way of operating the plant (*d*), by means of which the quality of the steam coal can be improved, is to recover as coking coal the portion of fraction A floating in a liquid of S.G. 1·34; rewash the refuse so as to recover the coal floating in a liquid of S.G. 1·38, screen the refuse from this operation through a $\frac{7}{8}$ in. mesh, crush and rewash. It may be seen from Table 32 that the bulk of the 1·38 to 1·425 float (12·84 per cent.) is larger than $\frac{7}{8}$ in. size, only a small proportion (1·84 per cent.) being smaller than $\frac{7}{8}$ in. Further, of the 1·425 sink (amounting to 9·13 per cent. of the total), 6·11 per cent. is over $\frac{7}{8}$ in. and only 3·02 per cent. is less than $\frac{7}{8}$ in. in size. Consequently, the plant may be operated to recover only the portion of the raw coal floating in a liquid of 1·38 S.G., the refuse may be screened through a $\frac{7}{8}$ in. screen, and that portion remaining on the screen may be crushed and rewashed, thus improving the yield. When this was tried experimentally, it was found that by crushing and rewashing the refuse greater than $\frac{7}{8}$ in. in size an additional recovery of 5·44 per cent. of the raw coal was made possible. The operating results in these circumstances are shown in Table 38.

The coking coal obtained by this method consists of 48·66 per cent. of the feed with 5·5 per cent. of ash and 0·97 per cent. of sulphur; 37·44 per cent. is recovered as steam coal with 10·87 per cent. of ash and 1·11 per cent. of sulphur.

The precise object to be achieved by the operator of the washing plant must be decided to a large extent by local conditions. If there is a good demand for coke and a poor demand for steam coal, either of the first two schemes would be preferable. If, however, there were a poor market for coke and a fair market for steam coal,

TABLE 38.—WASHING CHART FOR COKING COAL AND STEAM COAL. METHOD (*d*)

	Weight per cent. of Total Sample.	Ash per cent.	Sulphur per cent.
Coking coal—			
Fraction A, clean coal, 48·66 per cent. floating in 1·34	48·66	5·50	0·97
Steam coal—			
Fraction B, raw coal (– $\frac{3}{16}$ in.) . .	18·79	8·10	1·28
Fraction A, float in 1·38, sink in 1·34 + refuse recovery (13·21 + 5·44) . .	18·65	13·77	0·95
Resultant steam coal	37·44	10·87	1·11
Refuse	13·90	32·87	2·798
Original coal	100·00	11·34	1·28
Total yield . . .	86·1 per cent.		

one or the other of the second two schemes would be more profitable, especially since the coke produced under these schemes would be of high quality and would therefore be readily marketable.

In England it is not usual to collect several qualities of washed coal, but with a coal high in middlings the collection of a high quality coal and a second quality coal for boiler use is worthy of consideration.

3. *Interpretation of the Results of Practical Coal-cleaning Tests.*—If the examination of a coal prior to washing includes large scale tests on representative samples of the coal, the relative merits of several processes can be judged by float and sink tests on the products of washing. In all commercial processes, other than flotation processes, the raw coal is divided into fractions according to specific gravity. Thus, the washer may be adjusted to separate the raw coal into clean coal with a specific gravity less than, say, 1·5, and refuse with a specific gravity greater than 1·5. Actually the washer is set to produce coal with a certain maximum ash content, say, 5 per cent., and it is known that if all the particles of S.G. $< 1·5$ are included in the clean coal the ash content will be 5 per cent. The loss of particles of S.G. $< 1·5$ in the refuse reduces the yield of clean coal, and the presence of particles of S.G. $> 1·5$ in the clean coal increases its ash content.

If the washer is set to effect a separation at a specific gravity of 1·5, the clean coal and refuse produced in a washing test may be submitted to float and sink tests in a liquid of S.G. 1·5. The dirt not removed from the clean coal is recorded as "sinks in clean

coal," and the loss of useful coal as "floats in refuse." Any washer making a sharp differentiation at any given specific gravity can usually make an equally sharp separation at some other specific gravity. Although two washers may not have been set to make a separation at the same specific gravity, to compare the results of washing by the two processes, it is satisfactory to add together the figures for "sinks in clean coal" and "floats in refuse." The process by which this sum is a minimum may be regarded as the more efficient.

The proper interpretation of the float and sink analysis of the products of a coal-cleaning test enables an estimate to be formed of the amount of disintegration of the coal experienced in practice. The mechanical fracture of "pure" coal when subjected to a cleaning process increases the number of small particles and, under present conditions, decreases its market value. On the other hand, the disintegration of interstratified particles liberates coal and shale from the middlings, and therefore increases the yield of the marketable products.

In the ordinary way the results of float and sink tests on a sample of washed coal are recorded so that each fraction may be expressed as a percentage of the total sample of washed coal. A series of results, so recorded, is given in Table 39.

TABLE 39.—FLOAT AND SINK RESULTS ON WASHED COAL

Size (in.).	Specific Gravity.										
	< 1·3		1·3 – 1·4		1·4 – 1·5.		1·5 – 1·6		> 1·6.		
	Wt. per cent.	Ash per cent.	Wt. per cent	Ash per cent.	Wt. per cent.	Ash per cent.	Wt. per cent.	Ash per cent.	Wt. per cent.	Ash per cent.	Total Wt. per cent.
Over 1¾ .	20·2	2·9	10·5	7·8	5·0	12·8	2·0	25·6	1·2	45·2	38·9
1¾–1 . .	14·8	2·8	8·2	7·8	5·0	16·8	0·5	21·8	1·5	41·1	30·0
1–¾ . .	2·7	3·4	1·0	9·1	0·4	15·6	0·2	25·0		—	4·3
¾–½ . .	1·2	1·8	0·9	8·8	0·3	16·5	0·1	25·2	0·2	42·7	2·7
½–¼ . .	8·7	2·2	3·0	12·0	1·4	18·5	0·7	27·5	1·8	54·0	15·8
¼–0 . .	5·6	2·3	1·4	7·4	0·3	15·0	0·2	24·7	0·8	61·8	8·3
Total .	53·4	2·6	24·8	8·0	12·4	14·9	3·9	25·1	5·5	48·0	100·0

Similarly, the results of the examination of the refuse may be recorded, so that each fraction is expressed as a percentage of the total sample of refuse. If the weights of the washed coal and refuse are known, their relative proportions may be calculated. If the proportions are: washed coal n, refuse $1 - n$, the washed coal will constitute n of the raw coal and the refuse $1 - n$ of the raw coal. If each of the individual percentage weights of the washed coal in Table 39 be multiplied by n, the weights of each fraction of the washed coal are expressed as a percentage of the raw coal. Thus,

TABLE 38.—WASHING CHART FOR COKING COAL AND STEAM COAL. METHOD (*d*)

	Weight per cent. of Total Sample.	Ash per cent.	Sulphur per cent.
Coking coal—			
Fraction A, clean coal, 48·66 per cent. floating in 1·34	48·66	5·50	0·97
Steam coal—			
Fraction B, raw coal (− 1/16 in.)	18·79	8·10	1·28
Fraction A, float in 1·38, sink in 1·34 + refuse recovery (13·21 + 5·44)	18·65	13·77	0·95
Resultant steam coal	37·44	10·87	1·11
Refuse	13·90	32·87	2·798
Original coal	100·00	11·34	1·28
Total yield	86·1 per cent.		

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If the washer is set to effect a separation at a specific gravity of 1·5, the clean coal and refuse produced in a washing test may be submitted to float and sink tests in a liquid of S.G. 1·5. The dirt not removed from the clean coal is recorded as " sinks in clean

coal," and the loss of useful coal as "floats in refuse." Any washer making a sharp differentiation at any given specific gravity can usually make an equally sharp separation at some other specific gravity. Although two washers may not have been set to make a separation at the same specific gravity, to compare the results of washing by the two processes, it is satisfactory to add together the figures for "sinks in clean coal" and "floats in refuse." The process by which this sum is a minimum may be regarded as the more efficient.

The proper interpretation of the float and sink analysis of the products of a coal-cleaning test enables an estimate to be formed of the amount of disintegration of the coal experienced in practice. The mechanical fracture of "pure" coal when subjected to a cleaning process increases the number of small particles and, under present conditions, decreases its market value. On the other hand, the disintegration of interstratified particles liberates coal and shale from the middlings, and therefore increases the yield of the marketable products.

In the ordinary way the results of float and sink tests on a sample of washed coal are recorded so that each fraction may be expressed as a percentage of the total sample of washed coal. A series of results, so recorded, is given in Table 39.

TABLE 39.—FLOAT AND SINK RESULTS ON WASHED COAL

Size (in.).	Specific Gravity.										
	< 1·3.		1·3 – 1·4		1·4 – 1·5.		1·5 – 1·6		> 1·6.		
	Wt. per cent.	Ash per cent.	Wt. per cent.	Ash per cent.	Wt. per cent.	Ash per cent.	Wt. per cent.	Ash per cent.	Wt. per cent.	Ash per cent.	Total Wt. per cent.
Over 1¾	20·2	2·9	10·5	7·8	5·0	12·8	2·0	25·6	1·2	45·2	38·9
1¾–1	14·8	2·8	8·2	7·8	5·0	16·8	0·5	21·8	1·5	41·1	30·0
1–¾	2·7	3·4	1·0	9·1	0·4	15·6	0·2	25·0	—		4·3
¾–½	1·2	1·8	0·9	8·8	0·3	16·5	0·1	25·2	0·2	42·7	2·7
½–¼	8·7	2·2	3·0	12·0	1·4	18·5	0·7	27·5	1·8	54·0	15·8
¼–0	5·6	2·3	1·4	7·4	0·3	15·0	0·2	24·7	0·8	61·8	8·3
Total	53·4	2·6	24·8	8·0	12·4	14·9	3·9	25·1	5·5	48·0	100·0

Similarly, the results of the examination of the refuse may be recorded, so that each fraction is expressed as a percentage of the total sample of refuse. If the weights of the washed coal and refuse are known, their relative proportions may be calculated. If the proportions are: washed coal n, refuse $1 - n$, the washed coal will constitute n of the raw coal and the refuse $1 - n$ of the raw coal. If each of the individual percentage weights of the washed coal in Table 39 be multiplied by n, the weights of each fraction of the washed coal are expressed as a percentage of the raw coal. Thus,

in the test in which the figures in Table 39 were obtained $n = 0{\cdot}892$ and $1 - n = 0{\cdot}108$. The percentage of the raw coal greater in size than 1¾ in. which passes into the washed coal and floats at S.G. 1·3 is $20{\cdot}2 \times 0{\cdot}892 = 18{\cdot}0$.

Similarly, the percentage weights of each fraction of the refuse may be multiplied by $1 - n$ and be expressed as a percentage of the raw coal.

These calculations have been made, and from them it is possible to compute a table comparing the float and sink analyses of the raw coal and of a gross sample of the washery products (washed coal + refuse). For the test on this washery the comparison is drawn in Table 40.

TABLE 40.—COMPARISON OF RAW COAL AND WASHERY PRODUCTS

Weights as Per Cent. of Raw Coal and Products.

Size (in.).	Specific Gravity.										Total.	
	< 1·3		1·3 – 1·4		1·4 – 1·5		1·5 – 1·6		> 1·6			
	Raw Coal.	Products.	Raw Coal.	Products.	Raw Coal.	Products.	Raw Coal.	Products.	Raw Coal.	Products.	Raw Coal.	Products.
Over 1¾ .	18·4	18·0	6·7	9·2	4·8	4·4	1·8	1·8	7·2	2·5	38·9	35·9
1¾–1 .	16·4	13·1	6·5	7·2	2·1	4·4	1·2	0·4	6·2	3·8	32·4	28·9
1–¾ . .	6·2	2·4	3·8	0·3	2·7	0·4	0·6	0·2	3·2	2·2	16·5	6·0
¾–½ . .	5·0	1·1	1·6	0·8	0·5	0·3	0·2	0·1	1·3	1·4	8·6	3·7
½–¼ . .	1·6	7·8	0·3	2·8	0·2	1·3	0·1	0·7	0·2	4·4	2·4	17·0
¼–0 . .	0·6	5·0	0·2	1·3	0·1	0·5	0·1	0·2	0·2	1·7	1·2	8·5
Total .	48·2	47·4	19·1	22·1	10·4	11·1	4·0	3·4	18·3	15·9	100·0	100·0

From the figures in Table 40 it appears that, of the 38·9 per cent. of the raw coal greater than 1¾ in. in size, 3·0 per cent. is fractured during washing, and 10·5 per cent. of the raw coal between 1 and ¾ in. is broken into smaller sizes. The fracture which results causes the products to contain only 19·3 per cent. of material of specific gravity greater than 1·5, whereas the raw coal contains 22·3 per cent. heavier than 1·5. Obviously some of the heaviest particles have been broken up into dirt (which is still heavier than 1·5) and coal which appears amongst the washed coal. Because of this disintegration, the washer is recovering more "pure" coal particles than are actually supplied in the raw coal, these additional coal particles being concentrated in the fraction of S.G. 1·3 to 1·4. (Raw coal 19·1 per cent., washed coal 22·1 per cent.)

The fracture of the "pure" coal particles is indicated by the fact that the raw coal contains 48·2 per cent. lighter than S.G. 1·3, whereas the products contain only 47·4 per cent., 0·8 per cent. being broken during washing.

In this example of the method of determining the fracture of the coal and middlings during cleaning, the amounts of breakage are greater than is usually experienced. This is because the washer was arranged to collect a middlings fraction, which was crushed and returned for rewashing, but the example is given to illustrate the method and not as an instance of the amount of fracture that is usually encountered. With a friable coal, however, cleaned by a process in which there is a considerable mechanical stress on the particles, similar amounts of breakage may be experienced, but the fracture would be sustained by the lighter particles rather than by the heavier particles (as in this example).

If the results of screening tests on the raw coal, the washed coal, and the refuse were combined to compare the size distribution of the particles in the raw coal and in the washery products, the information obtained indicates the amount of fracture which has occurred, but it does not indicate whether the fracture has been sustained by the light coal particles, by the middlings particles, or by the dirt. No information is therefore obtained with regard to the decreased market value of the coal because of fracture of light particles, the increased yield because of fracture of the middlings, or the extent of the disintegration of the shale.

Washability Curves.—When the float and sink analysis of a coal has been performed, the results may be plotted graphically. They are then easier to understand, and are more instructive than when presented in tabular form.

The summarised float and sink analysis of a Yorkshire coal is given in Table 41. The observed values for percentages by weight and percentage ash contents are given in the first two columns, and in the last two columns the results are expressed on a cumulative basis.

TABLE 41.—FLOAT AND SINK ANALYSIS. YORKSHIRE COAL

S.G.	Weight per cent.	Ash per cent.	Cumulative Weight per cent.	Cumulative Ash per cent.
< 1·35	47·3	5·4	47·3	5·4
1·35 to 1·40	12·5	9·3	59·8	6·2
1·40 ,, 1·50	5·4	13·9	65·2	6·9
1·50 ,, 1·60	2·7	21·8	67·9	7·4
> 1·60	32·1	69·3	100·0	27·3
Total	100·0	27·3	—	—

The usual method of constructing the washability curve is to

plot the figures in the first two columns on squared paper, and a curve is obtained of weight against ash content. This is done in Fig. 2. The points A, B, C, D and E correspond to the figures in Table 41.

It will be observed that the point A is placed against the value 23·65 for the ordinate, or weight, and 5·4 for the abscissa, or ash content. In Table 41 the weight floating at a specific gravity of 1·3 is 47·3 per cent., and the mean ash content of this fraction is 5·4 per cent. The fraction is comprised of several qualities of coal, some lower in ash content than 5·4 per cent., some higher. As 5·4 is the mean value of the ash content, it may be assumed to correspond to the actual ash content of the average weight.

In plotting the curve, therefore, the weight corresponding to

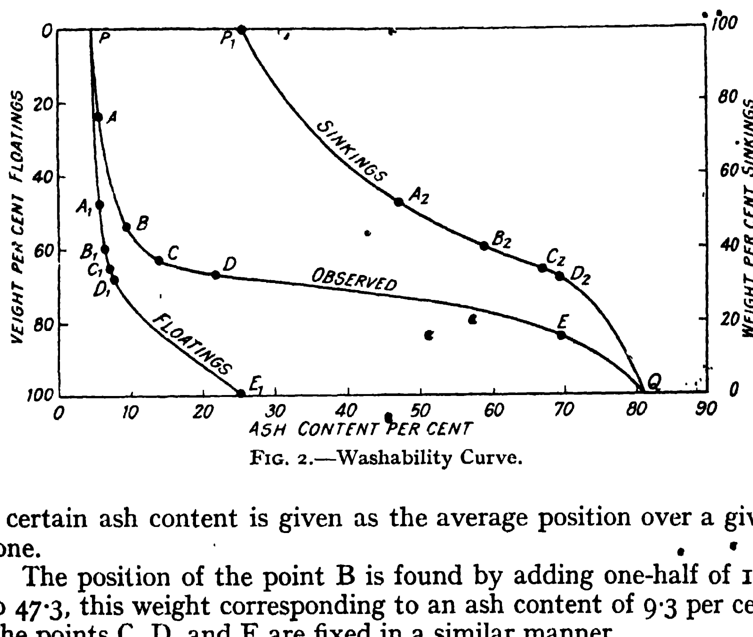


FIG. 2.—Washability Curve.

a certain ash content is given as the average position over a given zone.

The position of the point B is found by adding one-half of 12·5 to 47·3, this weight corresponding to an ash content of 9·3 per cent. The points C, D, and E are fixed in a similar manner.

These five points being fixed in this way, the curve is drawn and extrapolated to cut the axes at P and Q; the point P then corresponds to those particles of the raw coal containing the minimum ash content, and the point Q to those particles of shale with the highest ash content.

It will be observed that, as the curve approaches the axis for ash content, it is made to turn and cut the axis at a value of about 80. The position of this point Q is arbitrary, but usually it lies between about 75 and 85. Similarly, the position of the point P is arbitrary, for A, the nearest fixed point, falls some distance from the abscissa corresponding to zero weight. The position of these

points is, however, of no special value in the interpretation of the curves.

The curve P, A, B, C, D, E, Q being constructed and called the "Observed" curve, a second curve, P, A_1, B_1, C_1, D_1, E_1, is constructed from the figures in the last two columns of Table 41. This curve is the "floatings" curve, and the points on it are given by the figures of the cumulative weights and ash contents, these cumulative values being calculated by compounding the values for the fractions prepared in the float and sink tests. It therefore differs from the "Observed" curve in that no averages are introduced for the weights of the various fractions, the co-ordinates of the point A_1, for example, being 5·4, 47·3, as compared with 5·4, ½ (47·3) for the point A.

A third curve is then constructed, which may be called the "Sinkings" curve. This curve, P_1, A_2, B_2, C_2, D_2, Q, corresponds to the "Floatings" curve, but instead of being plotted from the fractions of the raw coal which float at different specific gravities, it is constructed from the fractions which sink. Like the floatings curve, it is based on cumulative figures, and relates the weights and ash contents of the sinking fractions.

Consider, for example, the results of testing the raw coal at a specific gravity of 1·35. At this specific gravity, 47·3 per cent. of the raw coal floats and the floating fraction contains, on the average, 5·4 per cent. of ash. The portion of the raw coal that sinks in a liquid of S.G. 1·35 comprises 52·7 per cent. of the total, and contains all the various fractions which subsequently float in liquids of S.G. 1·4, 1·5 and 1·6, and sink at 1·6. Its total ash content is therefore the sum of the ash contributed by each fraction, divided by their total weights (52·7). The ash content is therefore—

$$\frac{12{\cdot}5 \times 9{\cdot}3 + 5{\cdot}4 \times 13{\cdot}9 + 2{\cdot}7 \times 21{\cdot}8 + 32{\cdot}1 \times 69{\cdot}3}{52{\cdot}7} = 46{\cdot}9.$$

The point A_2 is therefore fixed by the values 52·7 for weight and 46·9 for ash content.

Similarly the co-ordinates of the point B_2 are;
Weight, 40·2.

$$\text{Ash content, } \frac{5{\cdot}4 \times 13{\cdot}9 + 2{\cdot}7 \times 21{\cdot}8 + 32{\cdot}1 \times 69{\cdot}3}{40{\cdot}2} = 58{\cdot}6.$$

The points C_2 and D_2 are fixed in a similar way, and the curve is drawn through these points and through the point Q on the "Observed" curve.* The uppermost point of the "Sinkings" curve corresponds to the use of a solution in which all the raw coal

* Generally, it is easier to fix the true position of the point Q from the "Sinkings" curve than from the "Observed" curve, and the position of the final point on the "Observed" curve is therefore usually left until the "Sinkings" curve has been constructed.

sinks (in the same way that the point E_1 corresponds to the use of a solution in which all the raw coal floats). In these circumstances the ash content of the sinkings fraction is equal to that of the raw coal (27·3 per cent.), and the point P_1 therefore falls on the same ordinate as the point E_1.

These three curves constitute the washability curves of the raw coal, and they are of the utmost importance in the examination of a coal for cleaning. From them one may tell at a glance the yield and nature of the products that can be obtained from the coal.

The "Floatings" curve is a cumulative curve, and the co-ordinates of any point upon it indicate the maximum theoretical yield of clean coal with a given ash content. Thus, if an ash content of 10 per cent. in the cleaned coal is satisfactory, the theoretical maximum yield of clean coal is shown, by the curve, to be 73 per cent. If it is desired to produce a clean coal with an ash content of 5 per cent., the maximum yield is only 42 per cent.

The "Sinkings" curve enables the theoretical ash content of the refuse to be determined with any given percentage recovery of the raw coal. Thus if the clean coal contains 10 per cent. of ash, at least 25 per cent. of the raw coal must be rejected as refuse, and it is impossible to obtain a refuse with a higher ash content than 74 per cent. Similarly, when the clean coal contains 5 per cent. of ash, the theoretical minimum yield of refuse is 58 per cent., with a maximum ash content of 45 per cent.

The coal for which the washability curves have been drawn in Fig. 2 is one containing a higher proportion of middlings (S.G. 1·35 to 1·6) than is commonly found in British coals. Usually about 60 to 90 per cent. of the raw coal floats at a specific gravity of 1·35, and the points from which the curve must be constructed are then grouped more closely together than is the case in the example considered. In these circumstances it is necessary to do float and sink tests at a wider range of specific gravities than 1·35 to 1·60, and specific gravities of 1·25, 1·3 are frequently employed in order to determine the co-ordinates of a larger number of points on the first portion of the curve. If, on the other hand, the coal contains a considerable proportion of material of higher specific gravity than 1·6 it may be advisable to do further tests in liquids of S.G. 1·7, 1·8, 2·0 or 2·2.

This is frequently necessary with continental coals and with anthracites. With this complete range of specific gravities it is not difficult to draw the curves accurately for the whole of their lengths.

It will be noticed that the "Observed" curve is drawn out towards the right after an ash content of 10 per cent. has been reached. This portion of the curve is usually of less importance than the portion representing ash contents below 10 per cent., and to magnify the scale in this more important portion, and to reduce the scale in the less important portion above an ash content of 10 per

cent., the curves are often plotted in logarithmic paper. The ordinates (weights) are usually maintained on a metric scale, only the ash contents being treated logarithmically.

The construction of washability curve in the customary manner has been described, but the usual form might with advantage be modified. There is little advantage (if any) in including the "Observed" curve, though its insertion appears to be the accepted rule. The "Observed" curve is inaccurate, for the exact value of the abscissa (weight) is arbitrary * and the curve yields no useful information. The "Floatings" and "Sinkings" curves are constructed from the float and sink results. They can be made accurate, and all the information required is supplied by them. It is true that there are methods of constructing the "Floatings" and "Sinkings" curves from the "Observed" curve, but the curves so constructed are liable to considerable errors, and the method offers no advantage over the method of calculation from the results of the float and sink tests. Moreover, only a small portion of the "Floatings" and "Sinkings" curves are usually required for use in connection with the examination of the coal. Under existing conditions, it is difficult to conceive of any purpose being fulfilled by washing the coal so that the yield is only 10 or 20 per cent. Seldom are those portions of the curves used which refer to yields of under 40 per cent. of the raw coal, and in these circumstances the upper four-tenths of the curves become useless. As has been stated, the majority of British coals contain over 60 per cent. of material floating at a specific gravity of 1·35, and only the lower four-tenths of the curves then serve any useful purpose.

If the curves are constructed so that only their useful portion is shown it is possible to employ a larger scale without increasing the area over which the lines are spread, and greater accuracy of construction and of reading are made possible.

The Henry Tube Method.—The construction of washability curves to determine the washability of a coal was first suggested by Charvet (*Bull.*, Soc. de l'Ind. Min., 2, 1903). The introduction of the Henry tube enabled washability curves to be constructed more easily than had previously been possible. The difficulty of obtaining accurate results with it, however, has led to its abandonment in many cases, and, on account of their greater reliability, float and sink tests are now adopted almost universally for the purpose.

The Henry apparatus consists of a tube 4 in. in diameter and about 2 ft. long. At the lower end of the tube a perforated metal plate is supported by piston rings. About 500 gm. of the coal are placed in the tube, being retained by the perforated plate. The coal sample tested should be freed from dust before placing it in the tube. The

* It will be remembered that the points are fixed by the mean ash content of given fractions and half their weights, as an approximation.

loss of dust through the perforated plate introduces an error, and the dust behaves irregularly, and its presence may interfere with the accurate interpretation of the results. The tube is immersed in water and moved rapidly up and down at least 50 or 100 times. During each downward stroke water passes through the perforated plate and agitates the coal contained in the tube. On the upward stroke the water drains away. The denser particles gradually accumulate at the bottom of the tube and a segregation according to density takes place.

When the agitation has been continued for a sufficient time, the particles are allowed to settle. The settled mass is then pushed up to the upper end of the tube on the perforated plate and is divided into sections by means of a knife. Each section is then dried, weighed and its ash content determined.

The top layer consists of the cleanest and lightest material, and each layer progressively downwards is more impure and heavier than the layer above it. For this reason it is necessary to cut the separated product into a relatively large number of layers (say, 10 or 12) each sufficiently thin to be properly representative. When each layer has been weighed, and its ash content determined, the results are used for the construction of washability curves. For this purpose, each layer may be regarded as a " float " fraction obtained by float and sink tests, and the results are employed in exactly the same way as the results of float and sink tests.

Separation by density, such as is effected by raising and lowering a Henry tube in water, is not exactly analogous to the separation effected during the operation of a jig or other washer. In using the Henry tube for the control of a jig it is therefore often preferable to place the tube at the bottom of the washing-box actually in use, and subject the sample for a comparatively long period of time to the movements of the water in the jig, rather than to agitate the tube in still water. Under these conditions, provision must be made to close both ends of the tube before it is withdrawn from the washing-box.

The accuracy of the Henry tube method of examining coal has been questioned by Maclaren (*Trans.* Inst. Min. Eng., 1925, 69, 315), and Sinnatt (*Journ.* Soc. Chem. Ind., 1927, 46, 249) records that it has been found necessary to jig the bed of material for one hour in order to effect stratification.

The X-ray Examination of Henry Tube Products.—To overcome the liability to error in using the Henry tube, Maclaren (*loc. cit.*) suggests that the stratified products in a Henry tube be examined by X-rays. An X-ray photograph of the contents of the tube is made, and in the photograph the coal appears as a semi-transparent image and the dirt gives an opaque image. The coal is jigged until the photograph indicates that the denser material has all been concentrated in the lower layers.

In view of the inaccuracy of the Henry tube method, and the

superiority of the float and sink method in every way (except perhaps in time), the use of X-rays for this purpose would appear to be expensive and unnecessary. Whilst of some scientific interest, the X-ray examination of coal in this connection would appear to be merely a complicated method of making a simple determination—that of ash content.

CHAPTER III

THE THEORY OF COAL WASHING IN JIG AND UPWARD-CURRENT WASHERS: GENERAL CONSIDERATIONS

It is impossible to design and control a jig or other washer scientifically and to appreciate the influence of the various factors concerned unless the fundamental principles underlying the process are understood.

There is probably no coal-cleaning process to which mathematical formulæ may be rigidly applied, but there is little doubt that the fundamental principle of the majority of coal-cleaning processes is as suggested by Rittinger sixty years ago. Rittinger's mathematical considerations (" Lehrbuch der Aufbereitungskundes," Berlin, 1867) led him to propound a formula for the terminal velocity of fall of a single solid in a liquid. This formula enabled calculations to be made on the speeds of currents of water required to separate certain particles from certain other particles. Although Rittinger's formula is now known to be incapable of explaining all the facts, later formulæ are almost all modifications of Rittinger's, with different values for the constants and with the inclusion of other terms or factors.

Although Rittinger's formula deals only with the fall of particles in water, by means of it, it is possible to draw conclusions which are of fundamental importance for coal-cleaning in jigs, in upward-current washers and on pneumatic tables, and there is little doubt that Rittinger correctly discovered the main fundamental principle involved in cleaning by these processes.

In this and the next chapter, Rittinger's mathematical theory is outlined and work subsequent to Rittinger's is described. Factors other than those considered by Rittinger are also examined. In the main, however, Rittinger's analysis of the motion of a particle in a body of liquid is followed. The theory of washing in currents of water which are substantially horizontal (trough washers and concentrating tables), and of cleaning by froth flotation, is given in the chapters dealing with these subjects.

The separation of raw coal into " pure " coal and refuse by means of a vertical current of water is effected through the difference in the specific gravity of the two portions of the raw coal. In a jig, the raw coal is placed in water which is periodically forced upwards and drawn downwards. The light coal particles subjected to the currents produced tend to ascend or remain near the surface; the heavy mineral particles tend to settle on the bed of the jig.

Motion of a Particle on a Still Liquid.—When a spherical particle of diameter, d, and of specific gravity, s, falls vertically in a liquid of specific gravity, σ, its effective weight (*i.e.*, the force producing acceleration) is its weight in air minus the weight of water it displaces and is equal to

$$\frac{\pi}{6}d^3w(s - \sigma)$$

where w is the weight of unit volume of water. When the particle is of irregular shape, the diameter, d, may be replaced by a value, r, where r is the diameter of the round screen hole through which the particle would pass, or is some other linear dimension of the particle.

If the liquid used is water, $w = \sigma = 1$ using suitable units; and the force is equal to $ar^3(s - 1)$ where a is a constant such that the volume of the particle is ar^3.

The resistance offered to the motion of the particle is proportional to the cross-sectional area which it presents to the liquid and to the square of its velocity. The area of cross-section equals br^2, where b is a constant, and the resistance equals

$$kbr^2v^2,$$

k being a further constant.

During the fall of the particle, the algebraic sum of the forces governing the motion is

$$ar^3(s - 1) - kbr^2v^2.$$

This force acts upon a mass ar^3s, and the general equation of motion may therefore be written,

$$\frac{ar^3s}{g} \cdot \frac{dv}{dt} = ar^3(s - 1) - kbr^2v^2 \quad . \quad . \quad . \quad . \quad . \quad (1)$$

Or,
$$\frac{1}{g}\frac{dv}{dt} = \frac{s-1}{s} - \frac{kb}{ars}v^2.$$

This equation reduces to $dt = \dfrac{1}{B}\dfrac{dv}{1 - A^2v^2}$

in which $A = \sqrt{\dfrac{kb}{ar(s-1)}}$ and $B = g. \dfrac{s-1}{s}$.

Solving this differential equation, a value for v may be obtained, viz.:—

$$v = \frac{1}{A} \cdot \frac{1 - e^{-2ABt}}{1 + e^{-2ABt}} + C$$

where C is the integration constant. Between the limits $v = 0$ and $v = v$, C becomes equal to zero, and the formula is

$$v = \frac{1}{A} \cdot \frac{1 - e^{-2ABt}}{1 + e^{-2ABt}} \quad . \quad . \quad . \quad . \quad . \quad . \quad (2)$$

When t is made infinitely long ($t = \infty$), the velocity v attains a limiting value $\frac{1}{A}$, for the term $\frac{1 - e^{-2ABt}}{1 + e^{-2ABt}}$ becomes equal to unity. Actually, as t increases from 0, v increases very rapidly and approaches its asymptotic value, $\sqrt{\frac{1}{k}\frac{a}{b}r(s-1)}$, after a very short interval of time, as may be seen from Fig. 3, which represents, in graphical form, the change in velocity with time.

The curve in Fig. 3 indicates that there are two phases of motion,

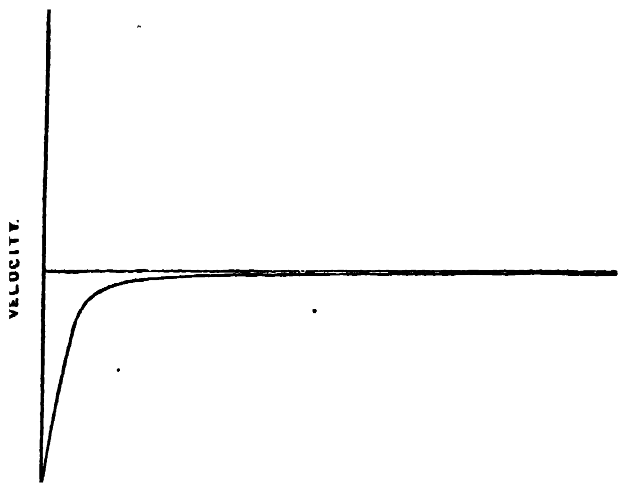


FIG. 3.—Increase in Velocity with Time. Particle falling i Liquid. Arbitrary Units.

namely, one of acceleration and one of substantially uniform motio.. During the initial stages of motion, $\frac{dv}{dt}$ is relatively great comparec with v, and in the value of $\frac{dv}{dt}$ (equation (1)), v^2 may be neglected The acceleration $\frac{dv}{dt}$ is therefore initially

$$\frac{dv}{dt} = \frac{g(s-1)}{}$$

This value of $\frac{dv}{dt}$ is independent of the linear dimension of the particle r, and of the constants a, b, k, and the motion of the particle

during the first stages of the initial acceleratory period is therefore independent of the shape and size of the particle, and is dependent solely on its density.

This principle is used in jigging operations, in which the rapid change in velocity and direction of the water current results in short falling times and small falling distances, with the result that a separation can be effected more or less according to density differences without a preliminary grading of the material according to size. Strictly speaking, the particles fall for only a brief interval of time in accordance with the equation :—

$$\frac{dv}{dt} = \frac{g(s - 1)}{s}$$

because, as soon as they begin to fall, the velocity increases rapidly and the term in v^2 (equation (1)) soon ceases to be negligible. There is therefore only a predominating tendency for separation according to density, irrespective of size. Actually, because of the influence of the velocity which the particles acquire, the acceleration decreases, until ultimately it becomes equal to zero, when t is infinite. With each increase in v, the influence of the size and shape of the particles comes more and more into play, and results in an increasing departure from the initial separation according to density differences. The acceleration becomes negligible, and v is a maximum, when the velocity reaches the value given by

$$\frac{kb}{ars} v^2 = \frac{s - 1}{s}$$

or,

$$v_{max.} = \sqrt{\frac{ar(s - 1)}{bk}} \quad . \quad . \quad . \quad . \quad . \quad . \quad (4)$$

The rate of fall during this period of negligible acceleration, or of approximately uniform motion, is characterised by the term $\sqrt{r(s - 1)}$, for the other factors under the root sign $\frac{a}{kb}$, are all constants. The equation may therefore be written

$$V = K\sqrt{r(s - 1)} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (5)$$

where $K = \sqrt{\frac{a}{kb}}$ and V is the terminal velocity of fall. This is the formula as first put forward by Rittinger.

Since $V^2 \alpha r(s - 1)$, it follows that the square of the final velocities of particles of the same density vary as their linear dimensions, and that the square of the final velocities of particles of the same size is a function of their density. Heavy large particles will therefore fall rapidly to the bottom, whereas small, light particles will fall least rapidly. Large light and heavy small particles will tend to fall together because of the compensation between size and density. Particles such as these are " like falling particles."

By equation (5),

$$V = K\sqrt{r(s - 1)}.$$

For two "like (or equal) falling particles,"

$$V_1 = K\sqrt{r_1(s_1 - 1)}, \quad V_2 = K\sqrt{r_2(s_2 - 1)}.$$

$$\text{But } V_1 = V_2\text{; therefore, } \frac{r_1}{r} = \frac{s - 1}{s_1 - 1}$$

These values apply only to the ultimate velocities of fall. The period during which "like falling particles" have the same velocity, is preceded by the acceleratory period, and during this, despite differences of shape and size, particles separate chiefly according to their density. Particles which cannot therefore be separated after attaining their terminal velocities can be separated before attaining them.

A rough rule for estimating the distance required to set up the final, or terminal velocity, as stated by Allen for a sphere (*Phil. Mag.*, 1900, 50, 324 and 519), is that the distance required for constant velocity to be attained is approximately five times the distance required to set up the same velocity *in vacuo*. The velocity V will therefore be attained after a fall of

$$h = \frac{5\,V^2}{2g} = \frac{V^2}{13} \text{ feet approximately.}$$

The ultimate velocity of fall in water of a rounded coal particle 1 in. in diameter is about 0·78 ft. per second, and according to Allen's rule its ultimate velocity will be attained after a fall of $\frac{(0{\cdot}78)^2}{13} = 0{\cdot}48$ ft. (or about 6 in.).

From this theoretical treatment of the vertical fall of particles in a still liquid, it will be seen that it might be possible under these conditions to effect a separation of the large, heavy particles from the small light particles, but that many of the particles of high specific gravity material would remain associated with those of low specific gravity material because of the compensating influence of size differences. Moreover, particles of very different sizes may remain together because of specific gravity compensations.

Motion of a Particle in Upward and Downward Currents of Water.—In order to achieve a separation of the coal and dirt particles, which, on account of the compensation between volume and specific gravity, are not separated by falling in a still liquid, impulses must be imparted to the liquid so that it is made to move periodically upwards and downwards. The particles may then be prevented from attaining their ultimate velocities, and periodically the motion of the particles corresponds to the period of acceleratory fall, during the initial stages of which separation by density may be effected. The influence of the motion of a particle of the motion

of the liquid in which it moves, is best investigated by a general consideration of the effect of a current of liquid on the motion of a particle on an inclined plane. Particular cases may then be considered.

Motion on an Inclined Plane.—In Fig. 4 the plane is inclined to the horizontal, the angle between the horizontal and the direction of motion of the water current being α. Let the velocity of the water along and parallel to the plane be W, and let μ be the coefficient of friction between the plane and the particle.

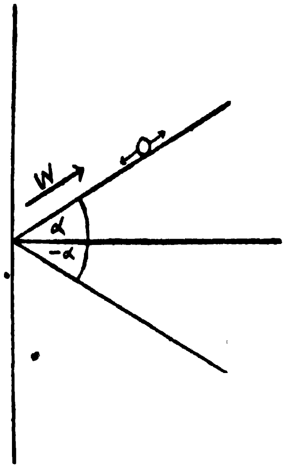


FIG. 4.—Motion of Particle on an Inclined Plane.

Resolving along the surface of the plane, the force acting on the body along the plane is equal to the algebraic sum of the force due to the current, the component of the weight of the particle parallel to the plane, and the frictional force opposing motion, *i.e.*,

$$kbr^2(W - v)^2 - ar^3 \sin\alpha(s - 1) \pm ar^3\mu \cos\alpha(s - 1).$$

The equation of motion may be written

$$\frac{ar^3s}{g} \cdot \frac{dv}{dt} = kbr^2(W - v)^2 - ar^3(s - 1)(\sin\alpha \pm \mu\cos\alpha).$$

$$\text{Or, } \frac{1}{g} \cdot \frac{s}{s - 1} \cdot \frac{dv}{dt} = \frac{kb}{ar(s - 1)}(W - v)^2 - (\sin\alpha \pm \mu\cos\alpha) \quad . \quad (6)$$

There are evidently two cases according as the quantity $(\sin\alpha \pm \mu\cos\alpha)$ is a positive or a negative quantity. In the first case, where $\sin\alpha \pm \mu\cos\alpha$ is positive, the water is flowing up the plane.

The differential equation may be solved as follows :—

As before, let $$\frac{kb}{ar(s - 1)} = A^2$$

and $$g.\frac{(s - 1)}{s} = B.$$

Let $\sin\alpha \pm \mu\cos\alpha = C.$

Then, $$\frac{1}{B} \cdot \frac{dv}{dt} = A^2(W - v)^2 - C.$$

Whence $$dt = \frac{1}{B} \cdot \frac{dv}{A^2(W - v)^2 - C}.$$

Putting $$\frac{C}{A^2} = D^2,$$

$$dt = \frac{dv}{BA^2 \cdot (W - v)^2 - D^2}$$

$$= \frac{1}{BA^2}\left[-\frac{1}{2D}\left\{\frac{dv}{W + D - v} - \frac{dv}{W - D - v}\right\}\right].$$

Integrating between the limits, $v = v$ and $v = 0$,

$$t = \frac{1}{2BA^2D} \cdot \log \frac{W + D - v}{W - D - v}.$$

Therefore, $$v = W - D\,\frac{e^{2A^2BDt} + 1}{e^{2A^2BDt} - 1}$$

$$= W - \sqrt{\frac{ar(s - 1)}{kb}}\,\sqrt{\sin\alpha \pm \mu \cos\alpha}\left(\frac{e^{2AB\sqrt{C}t} + 1}{e^{2AB\sqrt{C}t} - 1}\right).$$

In the limit, where $t = \infty$, the value of v becomes asymptotic and approaches the maximum value,

$$v_{max} = W - \sqrt{\frac{ar(s - 1)}{kb}}\sqrt{\sin\alpha \pm \mu\cos\alpha} \tag{7}$$

If the particle moves up the plane, with the water current,

$$v_{max} = W - \sqrt{\frac{ar(s - 1)}{kb}}\sqrt{\sin\alpha + \mu\cos\alpha} \quad . \quad . \quad . \quad . \tag{8}$$

If, however, the particle moves down the plane, against the water current,

$$-v_{max} = W - \sqrt{\frac{ar(s - 1)}{kb}}\sqrt{\sin\alpha - \mu\cos\alpha} \tag{9}$$

Motion in Vertically Upward Currents of Water.—Equations (6) and (7) define the motion of a particle on an inclined plane subjected to an upward current of water. By regarding the plane as being inclined vertically upward (90 degrees to the horizontal), the equations of motion in a vertically upward current of water may be deduced. Thus, by putting $\alpha = \frac{\pi}{2}$ in equation (7), $\sin\alpha \pm \mu\cos\alpha = +1$, and

$$v_{max} = W - \sqrt{\frac{ar(s - 1)}{kb}} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \tag{10}$$

which is the ultimate velocity with which a particle will ascend in an upward current.

It may be noted in passing that, for a horizontal current, $\alpha = 0$ and

$$v_{max} = W - \sqrt{\frac{\mu ar(s - 1)}{kb}} \tag{11}$$

During the initial stages of motion in a vertically upward current, for which $\alpha = \frac{\pi}{2}$, the particles move upward with an acceleration which, according to equation (6), is equal to

$$\frac{dv}{dt} = \frac{kbg(W - v)^2}{ars} - \frac{g(s - 1)}{s} \quad . \qquad (12)$$

It has been shown already that, when falling freely in a still liquid, a separation of two particles of different specific gravity takes place independently of their sizes during the first instant of their fall. During the first instant after they are subjected to an upward current, their acceleration is given by equation (12) as :—

$$\frac{dv}{dt} = \frac{kbg(W)^2}{ars} - \frac{g(s - 1)}{s} \quad . \; . \; . \; . \; . \; . \; . \; . \; . \; . \qquad (13)$$

by putting $v = 0$. As the velocity of the particle increases, its acceleration decreases, $(W)^2$ being replaced by $(W - v)^2$. The term

$$\frac{kbg(W - v)^2}{ars}$$

is a maximum for $v = 0$ and decreases as v increases. This is the term involving the size of the particles. The term $\frac{g(s - 1)}{s}$, which is independent of size, remains constant for all values of v. As v increases, therefore, the density factor has a greater influence than the size factor. It follows that there is a minimum tendency for separation in accordance with density differences, independently of size, when an upward current is first applied. Subsequently, as v increases, the tendency for separation according to density, irrespective of size, becomes greater. As v increases and the term involving the size of the particle decreases, the term $\frac{g(s - 1)}{s}$, which is independent of the size of the particles, and involves only their specific gravity, remains constant. It follows that as the particles accelerate, the separation is effected more and more according to density differences. For separation according to density differences only, and independently of size, it would be necessary for the term

$$\frac{kbg(W - v)^2}{ars}$$

to become equal to zero, that is when $W = v$. But equation (10) indicates that the maximum velocity the particles can attain in an upward current is less than W, so that v can never equal W. Consequently, separation strictly in accordance with density differences never takes place in an upward current of water. It is for this reason that upward current classifiers are unable to wash unsized coal.

It is known from experience that a Baum washer, for example,

can wash unsized coal. The reason why this is possible will be shown later. At present it is necessary to inquire more closely into the actual motion of particles when they are first subjected to an upward current.

Of two equal sized particles of specific gravity, s_1 (coal) and s_2 (dirt), a coal particle moves upwards initially more rapidly than a dirt particle, since from equation (12), if g_1 is the acceleration of the coal particle and g_2 that of the dirt particle,

$$g_1 - g_2 = \frac{dv_1}{dt} - \frac{dv_2}{dt},$$

v_1 and v_2 being the velocities of the coal and dirt particles respectively at any given instant of time. Putting $v_1 = v_2 = v$, as a close approximation at the beginning of the motion,

$$g_1 - g_2 = \frac{kbg(W - v)^2}{ar}\left(\frac{1}{s_1} - \frac{1}{s_2}\right) - g\left(-\frac{1}{s_1} + \frac{1}{s_2}\right)$$

$$= g\left(\frac{1}{s_1} - \frac{1}{s_2}\right)\left(1 + \frac{kb(W - v)^2}{ar}\right). \quad \quad (14)$$

Since $s_2 > s_1$, this expression is always positive, and g_1 greater than g_2. A coal particle, therefore, ascends more rapidly than a dirt particle of equal size.

Similarly, it may be shown that, if the upward current is of insufficient velocity to cause the two particles to ascend or to remain in suspension, the expression $g_1 - g_2$ is equal to

$$g\left(\frac{1}{s_2} - \frac{1}{s_1}\right)\left(1 + \frac{kb(W + v)^2}{ar}\right)$$

which, since $s_2 > s_1$, is a negative quantity. The dirt particle of higher specific gravity therefore falls against the upward current more rapidly than the coal particle. At the inception of an upward current, therefore, at whatever speed the current is moving, coal particles tend to attain or maintain a position above dirt particles of the same size.

The motion of two particles, one of coal and one of dirt, which are not of the same size may also be deduced from equation (12). For :—

$$g_1 = \frac{kbg}{ar_1s_1}(W - v_1)^2 - \frac{g(s_1 - 1)}{s_1}$$

and

$$g_2 = \frac{kbg}{ar_2s_2}(W - v_2)^2 - \frac{g(s_2 - 1)}{s_2}.$$

At the inception of the upward current, $v_1 = v_2 = 0$. Therefore,

$$g_1 - g_2 = \frac{kbgW^2}{a}\left\{\frac{1}{r_1s_1} - \frac{1}{r_2s_2}\right\} + g\left\{\frac{1}{s_1} - \frac{1}{s_2}\right\}. \quad \quad (15)$$

In order that a coal particle may move upwards more rapidly

than a dirt particle, g_1 must be greater than g_2. For this to occur it is necessary that

$$\frac{kbgW^2}{a} \cdot \frac{1}{r_1 s_1} + \frac{g}{s_1} \text{ be greater than } \frac{kbgW^2}{a} \cdot \frac{1}{r_2 s_2} + \frac{g}{s_2}$$

i.e., that

$$\frac{kbW^2}{a} \cdot \frac{1}{r_1 s_1} + \frac{1}{s_1} \text{ be greater than } \frac{kbW^2}{a} \cdot \frac{1}{r_2 s_2} + \frac{1}{s_2}.$$

But $\frac{1}{s_1}$ is always greater than $\frac{1}{s_2}$, since $s_2 > s_1$. The condition therefore is, roughly, that

$$\frac{1}{r_1 s_1} \text{ shall be equal to or greater than } \frac{1}{r_2 s_2}.$$

Equating these quantities,

$$r_1 s_1 \quad r_2 s_2$$

or
$$\frac{r_1}{r_2} = \frac{s_2}{s_1} \quad \ldots \ldots \ldots \quad (16)$$

If the specific gravity of coal is 1·3 and that of dirt 2·5, separation between the two will always take place continuously from the moment when they are first subjected to an upward current if the coal particle is $\frac{2 \cdot 5}{1 \cdot 3} = 2$ (approximately) times larger than the dirt particle. This suggests that sizing of the coal before washing in an upward current classifier must be very close, namely, of the order of 2 : 1. This apparent difficulty, or rather onerous requirement, is overcome in practice by suitable adjustment of the speed of the water current. It has been pointed out already that, as soon as the particles attain a velocity which is no longer negligible compared with the speed of the water current, the term in the equation of acceleration (equation (12)), involving the size of the particle, decreases, whilst that involving the specific gravity remains constant. As the velocity increases, therefore, the size ratio for separation at the beginning of motion becomes wider, and the necessity of a ratio of 2 : 1 no longer applies. Obviously, if the washer is so designed that the speed of the water current is low when it first meets the coal, the velocity of the particle relative to the water soon ceases to be negligible, and the size ratio widens very rapidly.

This is effected in the Draper washer by the use of a truncated cone-shaped washing chamber. The velocity of the water, whilst still able to force a coal particle upwards, is least at the top of the cone, where the coal enters, the velocity of the water at any cross-section increasing as the cross-sectional area decreases.

Another advantage of the low speed of current at the position of entry of the coal is seen from a closer examination of the argument

used to deduce equation (16). In course of the deduction, the terms $1/s_1$ and $1/s_2$ were removed, since s_2 is always greater than s_1. If, for s_1 and s_2, the values 1·3 and 2·6 be substituted, the condition for separation is that

$$\frac{kb}{a}W^2 \cdot \frac{1}{1\cdot 3r_1} + \frac{1}{1\cdot 3} > \frac{kb}{a}W^2 \cdot \frac{1}{2\cdot 6r_2} + \frac{1}{2\cdot 6},$$

or that,
$$\frac{2kbW^2}{ar_1} + 1 > \frac{kbW^2}{ar_2},$$

or that,
$$\frac{r_1 a}{2kbW^2} - \frac{r_1}{2r_2} + 1 > 0.$$

From this condition it follows that close sizing should be less necessary with large coal than with small coal, and, moreover, that separation will be facilitated if, at the beginning of the separation, W is small. The relative velocity of the current and the coal at the position where the raw coal enters should therefore be a minimum, and the coal should not fall from too great a height into the water. The speed of the current must, of course, always be sufficiently great to prevent the smallest coal particles from sinking. That is, it must be sufficient to keep $\frac{dv}{dt}$ (equation (13)) a positive quantity for the particles which float most easily, or

$$\frac{kb(W - v)^2}{ar_1} > (s_1 - 1).$$

This illustrates the advantage of using a conical separation chamber, as in the Draper washer. This washer will be described and its operation considered more fully in a later chapter. For the present it may be stated that the water current is so regulated that coal particles are not allowed to descend through the cone. At some position or other in the cone

$$\frac{kb(W - v)^2}{ar} > s_1 - 1$$

or, $W > K\sqrt{r_1(s_1 - 1)}$ since $\sqrt{\frac{a}{kb}} = K.$

On the other hand, dirt particles are able to sink through the cone, so that, for dirt particles,

$$K\sqrt{r_2(s_2 - 1)} > W.$$

The limiting ratio of sizes of particles which can just not be separated by making full use of the cone therefore becomes $\frac{s_2 - 1}{s_1 - 1}$, which is the size ratio for "equal falling" particles in a still liquid. It follows that, in an uqward current classifier,

particles of the same diameter but of different density will require current velocities in the ratio $\frac{W_1}{W_2} = \sqrt{\frac{s_1 - 1}{s_2 - 1}}$ to keep them in suspension. Particles of the same density but different size, will require velocities of currents in the ratio $\frac{W_1}{W_2} = \sqrt{\frac{r_1}{r_2}}$, and particles of different sizes and different densities will require currents of velocity ratio

$$\frac{W_1}{W_2} = \sqrt{\frac{r_1(s_1 - 1)}{r_2(s_2 - 1)}}.$$

Motion in a Vertically Downward Current of Water.—Before considering the principles upon which a jig washer acts, and particularly how, in certain cases, it is able to separate unsized coal and dirt, the motion of a particle in a downward current of water must be considered.

The motion of a particle under the influence of a vertically downward current may be examined as follows, the motion of a particle on an inclined plane being divided into two phases. At the beginning of the motion of the particle, its descent due to gravity is assisted by the current of water, and it therefore accelerates rapidly until its velocity is the same as that of the current. In the second phase, which begins when the velocity of the particle has become equal to that of the current, the acceleration of gravity causes the particle to fall more rapidly than the current of water, and its motion is consequently impeded by the resistance of the liquid. In each phase the effective weight of the particle is its mass less the mass of the volume of water displaced and equals $ar^3(s - 1)$.

The case is the second case of equation 6, $\sin\alpha \pm \mu\cos\alpha$ being negative. The particle always moves downwards and $\mu\cos\alpha$ has always the opposite sign to $\sin\alpha$. The plane will now be inclined with a negative angle equal to, say, α^1.

During the first phase,

$$\frac{1}{g} \cdot \frac{s}{s - 1} \cdot \frac{dv}{dt} = \frac{kb}{ar(s - 1)} (W - v)^2 + \sin\alpha^1 - \mu\cos\alpha^1 \quad . . \quad (17)$$

Using the notation previously employed, it may be shown that, when $v = W$,

$$t_{v=W} = \frac{1}{BA^2D} \tan^{-1} \frac{W}{D}.$$

During the second phase

$$\frac{1}{g} \cdot \frac{s}{s - 1} \cdot \frac{dv}{dt} = \sin\alpha^1 - \mu\cos\alpha^1 - \frac{kb}{ar(s - 1)} (v - W) \quad . \quad (18)$$

Whence,

$$t = \frac{}{2BA^2D}\left\{\log \frac{D + (v - W)}{D - (v - W)} + C\right\}.$$

Now write $t = t_{v=W} + t^1$, so that when $v = W$, $t^1 = 0$, and put

$$t^1 = \frac{1}{2BA^2D}\left\{\log \frac{D + (v - W)}{D - (v - W)} + C^1\right\}.$$

The general equation then becomes :—

$$t = \frac{1}{2BA^2D}\left\{\log \frac{D + (v - W)}{D - (v - W)}\right\} + t_{v-W},$$

whence

$$v = W + D\,\frac{e^{2BA^2Dt^1} - 1}{e^{2BA^2Dt^1} + 1}.$$

Putting $t^1 = \infty$,

$$\begin{aligned} v &= W + D \\ &= W + \sqrt{\frac{ar(s-1)}{kb}}\sqrt{\sin\alpha^1 - \mu\cos\alpha^1}. \end{aligned}$$

The particular solution is then obtained by putting $\alpha^1 = -\frac{\pi}{2}$, when $\sin\alpha^1 - \mu\cos\alpha^1 = +1$. The ultimate velocity of fall in a downward current is then found to be

$$V = W + \sqrt{\frac{ar(s-1)}{kb}}.$$

If in equation 17, α^1 be put equal to $-\frac{\pi}{2}$, the acceleration during the first phase is given by the relation :

$$\frac{1}{g}\frac{dv}{dt} = \frac{s-1}{s} + \frac{kb(W-v)^2}{ars}$$

At the beginning of the first phase, v is negligible and the acceleration of the particle is equal to

$$\frac{g(s-1)}{s} + \frac{kgb}{ars}W^2.$$

Two particles of the same size, but of different densities, s_1 (coal) and s_2 (dirt), will accelerate according to the equations :—

$$g_1 = \frac{g(s_1-1)}{s} + \frac{kgb}{ars_1}W^2, \text{ for coal,}$$

and

$$g_2 = \frac{g(s_2-1)}{s} + \frac{kgb}{ars_2}W^2, \text{ for dirt.}$$

The difference between these rates of acceleration is given by the value

$$g_1 - g_2 = \frac{g(s_1 - 1)}{s_1} - \frac{g(s_2 - 1)}{s_2} + \frac{kgbW^2}{ar}\left\{\frac{1}{s_1} - \frac{1}{s_2}\right\}$$

$$= g\left\{\frac{1}{s_2} - \frac{1}{s_1}\right\} + \frac{kgbW^2}{ar}\left\{\frac{1}{s_1} - \frac{1}{s_2}\right\}$$

$$= g\left\{\frac{1}{s_1} - \frac{1}{s_2}\right\}\left\{\frac{kbW^2}{ar} - 1\right\}. \quad . \quad . \quad . \quad . \quad . \quad (19)$$

If s_2 is greater than s_1, as is always the case, the term $\frac{1}{s_1} - \frac{1}{s_2}$ will be positive, and the difference between the acceleration of the particles will be positive unless $\frac{kbW^2}{ar}$ is less than unity. If the velocity of the water (W) is considerable, this will not be so, and $\frac{kbW^2}{ar}$ will be greater than unity, and g_1 will therefore be greater than g_2. The particle of lower specific gravity (coal) will therefore fall more rapidly than the particle of higher specific gravity (dirt). But the object of the motion of the water in jigging is to cause the dirt particles to fall more rapidly than the coal particles. It is accordingly necessary so to arrange conditions as that the velocity of the water is as low as possible. The term $\frac{kbW^2}{ar} - 1$ will then be negative, and g_2 will be greater than g_1, or alternatively, the term $\frac{kbW^2}{ar} - 1$ will be reduced to a small quantity and the coal particles will only fall very slightly faster than the dirt particles. Evidently, then, it is possible to draw the conclusion that too high a velocity of the descending water (suction) opposes the separation of equal-sized particles. Furthermore, since a high velocity of the descending water results in a more rapid fall of a coal particle than of a dirt particle, it will cause a loss of coal in the refuse, and this is well known to be the case in practice.

The second phase of the fall begins when the particles attain the velocity of the water current. When this occurs $W - v = 0$.

The acceleration of the particles is then

$$g_1 = \frac{g(s_1 - 1)}{s_1}, \text{ for coal particles,}$$

and $$g_2 = \frac{g(s_2 - 1)}{s_2}, \text{ for dirt particles,}$$

and the denser particle (s_2) falls more rapidly than the lighter particle (s_1). When this acceleration comes into play, the velocities of the particles increase until they move more rapidly than the

descending water current. They therefore meet the resistance of the water. The accelerations then become,

$$g_1 = \frac{g(s_1 - 1)}{s_1} - \frac{kbg(v_1 - W)^2}{ars_1},$$

$$g_2 = \frac{g(s_2 - 1)}{s_2} - \frac{kbg(v_2 - W)^2}{ars_2}.$$

If $v_1 = v_2 = v, g_1 - g_2 = g\left\{\frac{1}{s_2} - \frac{1}{s_1}\right\} \left\{1 + \frac{kb(v - W)^2}{ar}\right\}$. (20)

Since $s_2 > s_1$, $g_1 - g_2$ is negative, and the heavier particle accordingly falls the more rapidly. The particles cease to accelerate and fall with uniform velocity when

$$\frac{g(s - 1)}{s} = \frac{kbg(v - W)^2}{ars} \quad . \quad . \quad . \quad . \quad . \quad . \quad (21)$$

and the ultimate velocity is, therefore,

$$v = W + \sqrt{\frac{ar(s - 1)}{kb}} \quad (22)$$

If a particle be moving upwards and the upward current of water be changed to a downward current, the upward velocity of the particle will be reduced, and at some instant the relative velocity of the particle and the water will be zero, *i.e.*, W will be equal to v, and $W - v$ will equal 0.

From equation (18), the acceleration will be given by the equation :—

$$\frac{1}{g} \cdot \frac{dv}{dt} = \frac{s - 1}{s}$$

Or,

$$\frac{dv}{dt} = \frac{g(s - 1)}{s} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (23)$$

The acceleration will therefore be momentarily independent of the size of the particles and dependent only on their specific gravity. Conditions will then be reproduced similar to those obtaining at the beginning of the fall of a particle in a still liquid, and the rate of acceleration, relative to the liquid, will be the same as initially in a still liquid.

Motion of a Particle in Alternate Upward and Downward Currents of Water : Separation in a Jig :—It is now possible to discuss the behaviour of particles when subjected to alternate upward and downward currents, as in a jig washer. Separation of coal from dirt particles in a jig is effected principally during the upward current of water (down-stroke) and when the direction of the current is changing from upward to downward. The advantage of certain jig washers over many other types of washer is that preliminary sizing of the feed coal can be either eliminated or reduced to a minimum. It has been shown that complete separation of coal from dirt is imposible in an upward current without preliminary sizing, but that by using a low velocity of current at the outset, separation without pre-

liminary sizing is facilitated. If then the current speed increases rapidly, the value $W - v$ increases and separation as nearly as possible according to density differences is effected (though is never actually achieved in an upward current). Provided that the current speed eventually exceeds the terminal velocity of fall of the largest coal particle, the maximum cleaning that can be achieved during the downstroke of the plunger will be effected. If this velocity is attained, *i.e.*, if

$$W > K\sqrt{r_1(s_1 - 1)}$$

where r_1 is the size of the largest coal particle, all the coal particles will move upward. On the other hand, only those dirt particles whose size is given by the equation $W = K\sqrt{r_2(s_2 - 1)}$, whence $\frac{r_1}{r_2} = \frac{s_2 - 1}{s_1 - 1}$, will move upwards with the coal. The remaining dirt particles will remain stationary on the screen.

The initial slow upward movement of the current, which is desirable, is not due to the use of an eccentric (or link motion, as in some early types of jig washers), but is obtained as a natural consequence of the fact that a downward current of water is changed into an upward current. Before the water can move upward its inertia must be overcome. The upward current at its outset is therefore slow. Once the inertia has been overcome, the upward current rapidly attains its maximum velocity.

At the end of the downstroke, the plunger moves upwards and the upward water current in the wash-box is replaced by a downward current. Whilst the vertical direction of the water current is changing, the water, as far as vertical motion is concerned, is momentarily still.

The velocity of the particles subjected to the alternate upward and downward currents of water is also varying; and the direction of their motion may periodically be upward or downward. If, at the moment that the current of water is changing from an upward to a downward current, the particles become still relatively to the liquid, or if at some other time during the motion the relative velocity of the particles and of the liquid becomes equal to zero, the conditions are produced, for a brief interval of time, during which separation strictly according to density and independently of the size of the particles can occur.

If the particles and the water both become momentarily still, the conditions of fall are as given in equation (3)

$$\frac{dv}{dt} = \frac{g(s - 1)}{s},$$

and if their relative velocity becomes zero, the acceleration is also

$$\frac{dv}{dt} = \frac{g(s - 1)}{s}$$

from equation (23).

During the brief interval of time that these conditions obtain, all the dirt particles mixed with the coal fall through a greater height than the coal particles. The repetition of this process a number of times as the coal moves forward along the screen gives a predominating tendency for classification of the coal independently of its shape and size, and makes possible, in a Baum washer for example, the omission of preliminary sizing so necessary with many other types of washer.

During the upstroke of the plunger a downward current of water takes place in the wash-box (the so-called "suction") and, since this is liable to undo the separation which has previously taken place it is desirable to reduce it to a minimum. This is accomplished by allowing much of the water which has been pushed upwards during the downstroke of the plunger to be discharged with the washed coal, the deficit of water being made up by supplying water below the plunger. In this way the displacement of water during the upward stroke of the plunger is compensated by the inrush of water, without all the water being drawn through the bed. The conditions in and above the bed in the wash-boxes are therefore not materially affected by a downward current passing through it.

SUMMARY

The following general conclusions may be drawn from the above theoretical treatment of the motion of particles in water :—

(A) *In Still Water.*

(i.) Large heavy particles ultimately fall more rapidly than small light particles in accordance with the equation :—

$$V = K\sqrt{r(s-1)}.$$

(ii.) A heavy particle will ultimately fall at the same speed as a larger but lighter particle, because of the compensation between size and density when the sizes are in the ratio :—

$$\frac{r_1}{r_2} = \frac{s_2 - 1}{s_1 - 1}.$$

The values r_1 and r_2 may be measured by the diameter of the screen mesh through which the particles pass. Coal of specific gravity 1·3 and shale of specific gravity 2·5, will fall at equal rates if the size of the particles is in the ratio $\frac{r_1}{r_2} = \frac{2{\cdot}5 - 1}{1{\cdot}3 - 1} = \frac{5}{1}$,—that is, if the coal particle is five times as large as the shale particle. If the coal particle is less than five times the size of the shale particle, it will fall more slowly.

(iii.) During the initial stages of fall, the particle of higher specific gravity falls more rapidly than the particle of lower specific gravity, independently of the size of the particles. At the very beginning of the fall the acceleration of any particle is equal to $\frac{g(s-1)}{s}$,

Coal particles of specific gravity 1·3 therefore commence to fall with an acceleration of $\frac{32(1\cdot3 - 1)}{1\cdot3}$ or about 0·74 ft. per second per second, and shale particles of specific gravity 2·5 accelerate at a rate of about 1·92 ft. per second per second.

(iv.) During jigging, the direction of motion of the current should be reversed frequently. As the upward current is replaced by a downward current, the water is momentarily still, and at other times during the motion the liquid and the particles become still relative to each other. The conditions are then reproduced under which separation takes place according to density differences and independently of the sizes of the particles.

(B) *In an Upward Current of Water.*

(i.) A particle will move upwards if

$$W > K\sqrt{r(s-1)}$$

and, by suitably regulating the speed of the current, the larger and heavier particles may be made to fall downwards against the current, leaving the smaller lighter particles in suspension.

(ii.) Particles such that $\frac{r_1}{r_2} = \frac{s_2 - 1}{s_1 - 1}$ cannot be separated in an upward-current classifier with any one velocity of current. Coal particles of specific gravity 1·3 cannot be separated from shale particles of specific gravity 2·5 unless the ratio of the diameters $\frac{r_1}{r_2}$ is less than 5 to 1, and the particles must accordingly be screened to limits within this ratio if they are to be separated.

(iii.) During the initial stages of motion, coal particles ascend more rapidly than dirt particles of the same size.

(iv.) A particle will ultimately fall if

$$W > K\sqrt{r(s-1)}.$$

If particles of coal and dirt both fall, the velocity of the upward current being insufficient to cause either of them to remain in suspension, a dirt particle falls more rapidly than a coal particle of the same size.

(v.) The ultimate velocity of fall of a particle against an upward current is equal to

$$V = K\sqrt{r(s-1)} - W.$$

If the particle rises, its ultimate velocity is

$$V = W - K\sqrt{r(s-1)}.$$

Similarly, a particle neither rises nor falls but remains in suspension if

$$W = K\sqrt{r(s-1)}.$$

(vi.) Classification by jigging is facilitated by frequent upward currents.

(vii.) In an upward current, the tendency for the separation of heavy and light particles is greatest when the particles attain their terminal velocities, not, as in a still liquid, at the first instant of fall.

(C) *In a Downward Current of Water.*

(i.) The ultimate velocity of fall of a particle is given by the equation

$$V = W + K\sqrt{r(s-1)}.$$

(ii.) Initially a coal particle falls more rapidly than a dirt particle of the same size. To overcome this effect it is necessary, in a jig, to use the minimum possible velocity of downward current.

(iii.) Coal and dirt particles both accelerate until they attain a velocity equal to that of the current of water. Meanwhile the coal particle tends to fall a greater distance. Subsequently the dirt particles fall more rapidly than the coal particles.

It may be seen, therefore, that, whether the particle falls in still water, or is subjected to an upward or downward current of water, the velocity it will ultimately attain is governed by the expression

$$K\sqrt{r(s-1)}.$$

CHAPTER IV

THE THEORY OF COAL WASHING—THE INFLUENCE OF PARTICULAR FACTORS

THE equations deduced in Chapter III are not directly applicable to any coal cleaning process; they are fundamental but not particular. The constant, K, has different values under different conditions, and many factors are present during the operation of washing, of which no account has been taken. Nevertheless, the theoretical treatment of the subject allows many conclusions to be drawn which are substantiated in practice, and though but few of the equations may be rigidly applicable under industrial conditions, there is little doubt that the theory underlying them is correct in principle. It requires many modifications, however, to enable it to represent correctly the influence of all the factors involved.

It is often asserted that the Rittinger theory fails because, contrary to theory, it is possible to separate a wide range of sizes of raw coal into clean coal and dirt by washing in a jig. It was shown in the previous chapter, however, that this view is erroneous, and that the separation, by means of alternate upward and downward currents, of particles of different specific gravities, and differing materially in size, follows from the theory. There are, however, other factors involved which enable a wider range of sizes to be separated than is suggested by the Rittinger formula alone. The chief of these factors is the fact that, when raw coal is washed, there are a mass of particles enclosed in a relatively small bulk of liquid instead of a few isolated particles in an ocean of fluid. Still other factors, however, tend to narrow the limits of sizes between which separation is possible. One of these factors is the variation of the value of the constant, K. Each of these influences is considered in this chapter, and certain general deductions from theory are made.

VALUES OF THE CONSTANT K

Values of the constant, K, which, in the notation used in the previous chapter, equals $\sqrt{\frac{a}{kb}}$, have been deduced by workers other than Rittinger. Stating the formula as $V = K\sqrt{r_1(s-1)}$, where r_1 is the average diameter in inches of the mesh of the screen through which the particles would pass, and v is measured in feet per second, Rittinger's values of K were:—

TABLE 42

VARIATION WITH SIZE AND NATURE OF PARTICLES OF THE CONSTANTS *a* AND *b* WHICH RELATE THE DIAMETER OF THE SCREEN HOLE THROUGH WHICH THE PARTICLES PASS, WITH THEIR VOLUME AND CROSS-SECTIONAL AREA RESPECTIVELY

Diameter of screen passed.		Material floating in liquid S.G. 1·3.				Material sinking in liquid S.G. 1·3, floating in S.G. 1·9.				Material sinking in liquid S.G. 1·9.			
Mm.	Approximate equivalent in inches.	Weight of particle passing screen, mgm.		Value of *a*.	Value of *b*.	Weight of particle passing screen, mgm.		Value of *a*.	Value of *b*.	Weight of particle passing screen, mgm.		Value of *a*.	Value of *b*.
		Spherical shape.	Assorted shapes.			Spherical shape.	Assorted shapes.			Spherical shape.	Assorted shapes.		
0–2½	0–$\frac{1}{10}$	10·5	2·0	0·100	0·260	12·0	2·30	0·100	0·260	19·6	3·1	0·083	0·227
2½–4	$\frac{1}{10}$–$\frac{5}{32}$	43·7	10·0	0·120	0·294	48·3	10·8	0·117	0·285	82·3	13·0	0·088	0·240
4–6	$\frac{5}{32}$–$\frac{1}{4}$	145·0	38·7	0·138	0·320	165·0	42·0	0·133	0·310	272·0	51·0	0·098	0·253
6–8	$\frac{1}{4}$–$\frac{5}{16}$	350·0	140·3	0·210	0·422	387·0	147·0	0·190	0·395	618·0	184·0	0·156	0·350
8–10	$\frac{5}{16}$–$\frac{3}{8}$	676·0	280·0	0·217	0·432	762·0	292·0	0·201	0·410	1,227·0	420·0	0·180	0·382
10–20	$\frac{3}{8}$–$\frac{3}{4}$	—	—	0·390	0·605	—	—	0·330	0·595	—	—	0·280	0·520
20–30	$\frac{3}{4}$–1¼	—	—	0·430	0·685	—	—	0·400	0·664	—	—	0·340	0·590
30–50	1¼–2	—	—	0·482	0·730	—	—	0·438	0·705	—	—	0·382	0·645

2·67 for spherical particles.
1·43 for rounded bodies.
1·00 for flattened bodies.
1·24 for elongated bodies.
1·28 as the average for particles of various shapes.

A large number of more recent researches have suggested different values for the constant, but as the actual value is of little other than academic interest it is not proposed to discuss them. Reference may, however, be made to one series of experiments by M. de Caux.

M. de Caux (*Ann. des Mines de Belg.*, 1921, 22, 1103), from a study of a Belgian raw coal, determined for particles of different shapes and sizes the values of the constants a and b, which were introduced in Chapter III to correlate the volume and area of cross-section of particles with the diameter of the screen hole through which they pass. For a full description of the method adopted the original paper should be consulted. It consisted, briefly, of calculations based on a determination of the weights of particles of spherical and of assorted shapes, passing through given sieves. He reached the conclusion that the assumption made by Rittinger and others that the value of K is constant for particles of the same shape but of different sizes is untenable. His figures for the variation of the constants a and b are given in Table 42. Values for the constants are given with respect to the size not only of coal particles but also of dirt particles. Rittinger (and others) assume that the constant K has a uniform value for both coal and dirt.

From the results recorded in Table 42, the value of the constant, K, for raw coal material of different specific gravities may be tabulated as follows, k being given the value 0·0715 :—

TABLE 43

VALUE OF CONSTANT, K, IN FORMULA $V = K\sqrt{r(s-1)}$
(V MEASURED IN M. PER SECOND, r IN M.)

Dia. of screen Mm.	Coal. S.G. 1·3 K_c.	Middlings S.G. 1·3–1·9. K_m.	Dirt. S.G. 1·9. K_d.
0–2½	2·32	2·32	2·26
2½–4	2·39	2·39	2·27
4–6	2·44	2·44	2·33
6–8	2·63	2·60	2·50
8–10	2·68	2·64	2·56
10–20	2·84	2·77	2·73
20–30	2·97	2·91	2·83
30–50	3·04	2·95	2·88

The results obtained by de Caux, and reproduced in Table 43,

refer to particle sizes reduced to metres and velocities measured in metres per second. The values of these constants for particles measured in inches and for velocities in feet per second are given in Table 44, and graphically in Fig. 5. M. de Caux considers that the difference between the values of K for different particles is too

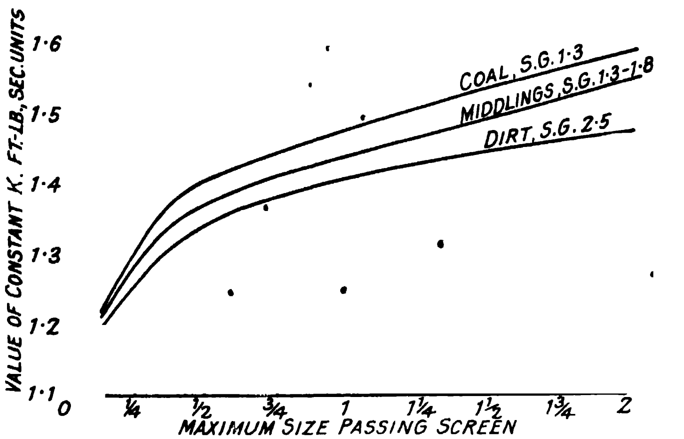


FIG. 5.—Values of Constant K for Particles of Different Sizes and Specific Gravities.

great to permit the use of any one value as a general value for all grades of material.

TABLE 44

VALUE OF K IN FORMULA $V = K\sqrt{r}\,(s-1)$ (V MEASURED IN FEET PER SECOND, r IN INCHES)

Dia. of screen. Inches. r.	Coal S.G. < 1·3 K_c.	Middlings S.G. 1·3–1·9 K_m.	Dirt S.G. > 1·9. K_d.
0–⅛	1·22	1·22	1·20
⅛–¼	1·30	1·29	1·26
¼–⅜	1·38	1·35	1·31
⅜–½	1·41	1·38	1·35
½–⅝	1·43	1·40	1·37
⅝–¾	1·45	1·42	1·39
¾–1	1·49	1·45	1·42
1–1¼	1·52	1·48	1·44
1¼–1½	1·55	1·51	1·46
1½–2	1·60	1·56	1·49

M. de Caux therefore suggests that the values given by Rittinger and other, more modern, writers need revision in two respects.

Firstly, different constants are required for coal, middlings and dirt; secondly, for each of these three types of material, different constants are required for different sizes.

These values of K may be substituted in the Rittinger formula for the terminal velocity of fall of particles in a liquid. A particle of coal of S.G. 1·3 and 2 in. in size will require a current of velocity $V_1 = 1{\cdot}6\sqrt{2(1{\cdot}3 - 1)} = 1{\cdot}240$ ft. per second to support it. Similarly, a particle of shale of S.G. 2·5 and $\frac{9}{16}$ in. in size will require a current of velocity $V_2 = 1{\cdot}36\sqrt{0{\cdot}562(2{\cdot}5 - 1)} = 1{\cdot}247$.

According to these values, the theoretical ratio for sizing before washing should be

$$\frac{r_1}{r_2} = \frac{2}{0{\cdot}562} = 3{\cdot}6$$

instead of 5, the value calculated from the expression

$$\frac{r_1}{r_2} = \frac{s_2 - 1}{s_1 - 1}.$$

The differing values determined by M. de Caux for different particles of a Belgian raw coal are possibly due to some phenomenon associated with the fracture of the coal. Whereas one coal, when broken up, may give particles which do not differ widely in shape, another coal may fracture into particles the shape of which shows no tendency to regularity. In view of this fact, it would appear desirable, in applying the theory to the washing of any particular coal, to examine its mode of fracture with some care and to determine, for that coal, what value or values of K are applicable.

THE INFLUENCE OF OTHER FACTORS

The formulæ considered hitherto are deduced for a single particle falling freely in a liquid medium. It would appear that when, as in coal washing, the raw coal is fed in bulk, not particle by particle, other factors come into play. Whereas for a single particle falling in an ocean of fluid the actual displacement of water is negligible, when a mass of coal is fed into the restricted space of a washing box, the water displaced may attain considerable proportions and its displacement will result in upward eddy currents around the particles. Furthermore, in order to effect a separation of the lighter from the denser material, the bulk of particles must be separated into individual units and some of the mechanical force which would otherwise be spent in imparting given velocities to the individual particles is used up, partly in effecting a mechanical separation of the aggregate, and partly in overcoming the frictional forces between the particles as they slide over and around each other. A further departure from simple theoretical expectations is provided by such other influences as the shielding or trapping of a small light particle on the under surface of a large heavy particle

and also by reason of the fact that, when the mechanical forces involved come into play, neither the water nor the particle begins to move, of necessity, from rest. The coal may be shot into the washer, the liquid medium may be subject to eddy and vortex currents, and consequently the motion of the particles may be irregular and not vertical.

Hindered Settling.—The mass effect results in the creation of forces which give rise to the phenomenon of "hindered settling." Munroe (*Trans.*, Amer. Inst. Min. Eng., 1888, 17, 637), who first investigated the phenomenon, showed that a sphere falls more slowly in a narrow tube than in a wide one, and, as a result of his experimental investigations, he put forward values for the constant K, which differ from those of Rittinger, as follows :—

Shape of Particle.	Value of K	
	Munroe Hindered settling.	Rittinger. Free. fall.
Small spherical	0·432	2·67
Angular, uniform size	0·278	1·28
Rounded	0·255	1·43
Large spherical in mass of small spheres .	0·160	—

Munroe's experiments enabled him to put forward the formula :—

$$V - V \cdot \frac{D^2 - r^2}{D^2}$$

where V_1 is the observed velocity of fall in a cylinder of diameter D, and V_2 the velocity of fall in an unrestricted vessel, r being the diameter of the particle.

Conclusions similar to those of Munroe were reached by Ladenburg (*Wied. Ann. der Phys.*, 1907, 41, 22, 287). Apart from the researches of Munroe and of Ladenburg, however, experimental investigation of the phenomenon of hindered settling appears to have been neglected, with the result that knowledge of the laws of hindered settling, and the influence of the phenomenon on the normal laws of fall, are quite indefinite.

In America, however, Fahrenwald (*Min. and Met.*, 1926, 7, 437) is conducting a series of experiments on hindered settling, and his results should add materially to our knowledge of the subject.

Louis, in a recent paper (*Chem. and Ind.*, 46, 545), has proposed a theoretical formula for the terminal velocity attained by a mass of spheres, all of the same size, falling in a body of fluid.

He assumes that the mass of the fluid remains constant, but that the fluid displaced by the falling spheres adds to the buoyant effect of the medium, acting, in fact, as though it increased the density of the medium.

Louis' formula for the terminal velocity of fall of the mass of spheres introduces a term m, the coefficient of mass density. The formula may be written :—

$$V = K\sqrt{\frac{r}{\sigma}\{s - \sigma(1 + m)\}}$$

using our notation. Putting $\sigma = 1$ for water,

$$V = K\sqrt{r(s - 1 - m)}.$$

For any given set of conditions m is a constant, depending for its value upon the mass of the spheres and the bulk of the fluid.

Besides the buoyant effect of the fluid displaced by a mass of spheres falling in a limited bulk of the fluid, other factors are concerned in the phenomenon of hindered settling when it is applied to the conditions obtaining in coal washing by a jig or an upward current classifier. The particles are then of various sizes and various specific gravities and their relative positions in the mass vary at different instants of time. The influence of friction between the particles is therefore introduced, eddy currents arise, and collisions between particles affect their relative directions and velocities of movement. The significance of these effects is not, at present, understood, and is still less capable of mathematical examination.

Later in the chapter the influence of friction between the particles is tentatively examined.

It is possible, however, that the influence of hindered settling on the working of a modern jig is not very great, except for the effect of mass density, the factor considered by Louis as adding to the buoyancy of the liquid medium. The pulsations of the liquid are so frequent and rapid that, even though at first they resulted in the performance of little useful work other than the separation of the particles of the mass, subsequently, sufficient time and opportunity is allowed for the normal laws of fall (with a modification similar to that of Louis) to become operative.

In an upward current classifier, however (*i.e.*, in the absence of pulsation), it might reasonably be expected that considerable power would be required to overcome the frictional forces between the particles and to effect an efficient separation between them. The phenomenon would, of course, have a considerable influence if the separation were to be effected solely by allowing a dense mass of particles to fall through a still liquid contained in a narrow chamber, but such a process is not a practicable proposition. Consequently, the true phenomenon of hindered settling, as defined by Munroe, hardly arises in washing practice, except in truly hindered settling classifiers (*e.g.*, the Richards). In most other washers,

external mechanical forces are employed which overcome, perhaps almost completely, the influence of the other factors, which, though they are not strictly included in the phenomenon of hindered settling, are usually grouped with it. To what extent such external mechanical forces are required is unknown. The final design of a washer is largely a matter of trial and error, and little effort seems to have been made to define in exact terms all the scientific laws applicable to washery practice and to design washers by their systematic application. A design based mainly on experiment must include a substantial margin for unknown factors. If these factors were studied and understood it might occur that much of the usual margin is unnecessary and that slight modifications of the plant, or of its method of operation, would themselves provide all the margin required.

Formation of Slimes.—Very small particles do not obey the laws which have already been deduced for the fall of particles in water. Since their behaviour is problematical, no washer depending upon density differences has been designed to deal successfully with very fine coal.

It will be recalled that, in obtaining the equation of motion of a particle falling in a fluid at rest, it was assumed that the resistance of the fluid to the fall was proportional to the square of the velocity of fall. By the principles of dynamic similarity, it may be shown that the resistance experienced by any body in motion through a fluid may be expressed as :—

$$R = \rho l^2 v^2 \, . \, f\left(\frac{vl}{\nu}\right)$$

in which ρ is the density of the fluid, l is a linear dimension of the body, v its velocity and ν the kinematic coefficient of viscosity of the fluid. The classical experiments of Froude (Brit. Ass. Rep., 1872) showed that, for considerable velocities, the resistance varied as v^2 and that $f\left(\frac{vl}{\nu}\right)$ was a constant term, the velocity being, therefore, independent of the viscosity of the medium. This conclusion has been substantially confirmed by numerous other workers, *e.g.*, Unwin (*Proc.* Inst. C. E., 1884, 80, 221), Allen (*Phil. Mag.*, 1900, 50, 324), Martin (*Trans.* Inst. Chem. Eng., 1926, 4, 164), and others, who have found that the power to which v must be raised is very nearly 2. The resistance to a particle falling in water with a considerable velocity may therefore be written $R = kl^2v^2$, which is the form employed at the beginning of this article.

When, however, the velocity of the particle is very small,* the resistance varies as the velocity, and the term $f\left(\frac{vl}{\nu}\right)$ is no longer,

* For water, a velocity of not more than 1 in. per sec. is specified (*Encyclopædia Brit.*, Hydraulics, s. vii). Considerable velocities are velocities greater than 1 in. per sec.

therefore, a constant.. For spheres, moving under these conditions, the ultimate velocity of fall may be calculated according to Stokes' law, as follows :—

$$V = \frac{2}{9} g r^2 \frac{s - \rho}{\nu \rho}$$

in which ν and ρ are the kinematic coefficient of viscosity and the density respectively of the fluid, r and s being the radius and specific gravity respectively of the sphere.

For considerable velocities and particles of considerable size, the motion of the particles is independent of the viscosity of the medium, the energy expended by the resistance of the fluid being used up in creating turbulent motion in the fluid around the particle. With very small particles, however, the stream line form of the fluid is not disturbed and the motion of the particle is hindered in overcoming the viscous resistance of the medium in which it moves.

Richards (" Text-book of Ore Dressing," Vols. 1 and 2, New York, 1903 ; Vols. 3 and 4, 1909) conducted an extensive investigation of the fall of single particles in water, and his experiments led him to the conclusion that the Rittinger theory, though applicable in general to relatively large particles, was no longer applicable below a certain size of particle, the size varying according to the nature of the material used. Below the critical size, the velocity of fall corresponded to the formula

$$V = K_1(s - 1)r^2$$

which is similar to the Stokes' law, the constant terms of Stokes and the viscosity of the fluid being included in the composite constant K_1. Richards found, moreover, that the change from one formula to the other occurred abruptly.

A formula for the terminal velocity acquired by very small particles was put forward by Wagoner. Wagoner (Ass. Eng Soc., 1896, 17, 73) used Richards' experimental values, and found that the velocity could be written

$$V \qquad \frac{D^{2}}{\sqrt{m}D^2 + n}$$

where c, m and n were constants. One interesting, but unexplained, point discovered by Wagoner was that $m + n = \sqrt{2}$.

Cooper and Wilson (*Coll. Guard.*, 1923, 126, 147) have also shown that particles of minerals falling in water obey different laws according to whether they are large or small, and found that the change from one law to the other occurred abruptly at about 50 mesh size, whatever the composition of the particles.

It is well known in practice that, in washing a raw coal with which the fines are included, the smallest particles which ultimately form slimes do not obey the laws governing the larger particles, but remain in suspension in the water. Many clays disintegrate in

contact with water, and the clay particles thus dispersed also form a suspensoid in the water. The effect of this suspension of coal and dirt particles is threefold. Firstly, because of the distribution of the fine particles through the washing water, its effective density is raised and instead of being unity becomes, say, 1·1. Lincoln (*Bull.* 69, Univ. of Ill., 23) considers that, in some cases, the density of the washing water is as high as 1·15 on account of the suspended impurity. Secondly, much of the fine coal is discharged with the washing water, from which it must be recovered. Thirdly, the finest particles of dirt are prevented from falling with the larger dirt particles, and some of them therefore pass over with the clean coal discharged. This dirt remains in the cleaned coal in the drainage hopper unless it is previously removed by spraying the coal with water on a screen or drainage conveyor.

Size of Particles Forming Slimes.—It is desirable to have some information with respect to the minimum size of particle that will obey the normal laws of fall, and consequently the extent to which fines must be removed from the raw coal to prevent the formation of slurry. According to H. S. Allen (*Phil. Mag.*, 1900, 50, 324 and 519), Stokes' law no longer holds good for spherical particles with a critical radius given by the expression :—

$$r = \sqrt[3]{\frac{9\nu^2}{2g\rho(s - \rho)}}$$

using the nomenclature previously employed.

Taking $\nu = 0{\cdot}0115$ c.g.s. unit for water at 15° C., for a coal particle of specific gravity 1·3 and a shale particle of specific gravity 2·5, the critical radii are, for coal 0·012 cm. (= 0·0041 in.) and for shale, 0·0047 cm. (= 0·0019 in.). By experiment, Cooper and Wilson (*loc. cit.*) determined the critical diameter as being about 0·01 in.* for a number of different substances, including coal and dirt. Von Jungst (*Glückauf,* 1914, 50, 6) gave the smallest size of coal that can be settled in water as varying from 0·02 in. to 0·008 in.

These figures indicate that only those particles of smaller size than about 0·02 in. fail to obey the ordinary laws of fall, and, consequently, only particles smaller than 0·02 in. ($\frac{1}{50}$ in.) should form slimes. In practice a $\frac{1}{50}$ in. screen requires a considerable ground space, and is difficult to operate (especially if the coal is not quite dry), and it is more convenient to remove all the coal passing through, say, a $\frac{1}{16}$ in. screen, or to remove the dust by elutriation or by aspiration. Even then slime formation is not eliminated, for some fine dust will remain in the coal and more fine dust will be produced by the mechanical stresses arising in transporting the coal to the washer and in washing.

It should be observed that the formulæ dealing with the motion of fine particles are applicable only to spherical particles, whereas it

* Passing 50-mesh screen. Standard screen size not specified.

is known that the shape of coal dust particles is very irregular. The subject of the influence of shape on the behaviour of particles during fall is one of considerable, but as yet undetermined, importance.

It should be recalled that, when considering the fall of the particles of such a size that they are unaffected by the viscosity of the medium, two constants were introduced, a and b, the former being a volume coefficient, the second a cross-sectional area coefficient. These two constants might, perhaps, be unnecessary if the linear dimension r were referred to the specific surface * of the particles, and not to the radius of the equivalent sphere or to the diameter of the screen hole through which the particles pass. The specific surface of a body may be defined as its outer surface area divided by its weight. It is on account of variations in the constants a and b with different sizes and shapes of particle, that de Caux found and introduced varying values of the composite term

$$K = \sqrt{\frac{a}{kb}}.$$

Washing in Dirty Water with an Effective Specific Gravity greater than that of Clean Water.—If it be assumed that the total amount of solid material in suspension is sufficient to raise the specific gravity of the washing water to 1·1, the formulæ which have been deduced for the velocity of particles under different conditions require modification. The conclusions drawn from these deduced formulæ do not, however, require revision ; the effect upon them is one of degree only. The modification suggested by Lincoln is as follows :—

For the ultimate velocity of fall of a particle, the value

$$V = K\sqrt{\frac{1}{1 \cdot 1}r(s - 1 \cdot 1)}$$

must be substituted for the formula $V = K\sqrt{r(s - 1)}$. Similarly, the linear ratios of equal falling particles are

$$\frac{r_1}{r_2} = \frac{s_2 - 1 \cdot 1}{s_1 - 1 \cdot 1} \text{ instead of } = \frac{s_2 - 1}{s_1 - 1}.$$

For coal of specific gravity 1·3 and shale of specific gravity 2·5,

$$\frac{r_1}{r_2} = \frac{1 \cdot 4}{0 \cdot 2} = \frac{7}{1}$$

in a liquid of which the effective specific gravity is increased to 1·1 by the suspended solids. It has previously been stated that, in pure water, the specific gravity of which is 1,

$$\frac{r_1}{r_2} = \frac{1 \cdot 5}{0 \cdot 3} = \frac{5}{1}.$$

The Chance washer depends for its action upon the increase in

* See Williams, *Trans.* Faraday Soc., 1922, 18, 87.

density of the washing water by the suspension therein of a finely divided solid. If, in this process, the specific gravity is increased to, say, 1·3, the ratio $\frac{r_1}{r_2}$ becomes equal to $\frac{1\cdot 2}{0}$, so that preliminary sizing is unnecessary.

It therefore follows that an increase in the specific gravity of the washing water by the suspension therein of fine particles increases the size ratio of equal falling particles. It should, therefore, increase the ease of separation of coal from dirt. For, whereas in practice it is not difficult to separate large coal particles from large dirt particles, or medium-sized coal particles from medium-sized dirt particles, it is relatively difficult to effect a complete separation of large coal particles from small dirt particles, especially when the sizes are such that the particles have equal terminal velocities.

The widening of the size ratio of equal falling particles by the use of a medium of higher specific gravity than unity should reduce the chances of equal-falling particles of coal and dirt being present in a sample of raw coal fed to a washer, after sizing between two limits. For example, if the feed coal is screened between the limits $\frac{3}{8}$ in. and 2 in. and the size ratio of equal-falling particles is 5 to 1 (as with water S.G. 1), there is no separation between $\frac{3}{8}$ in. dirt and $1\frac{7}{8}$ in. coal. If, however, the size ratio of equal-falling particles is 7 to 1, then $\frac{3}{8}$ in. dirt would be separable from all coal smaller than $2\frac{5}{8}$ in.

A washery manager always prefers to use clean water, since he usually finds that more satisfactory cleaning is thereby obtained in practice. These two statements (one, from theory, that water of higher density than unity should aid the separation of those particles which are most difficult to separate; and the other, from practice, that separation is not aided) are conflicting and require reconciling. The first explanation that suggests itself is that the extra ash content of a washery product when washed in dirty water is due to the film of dirt around each particle. This explanation, however, is not sufficient to account for the increase in ash content sometimes found in practice. It is not, therefore, capable of explaining the facts entirely. It does, no doubt, have some psychological influence, for the dirt film destroys the clean lustrous surface of wet coal washed in clean water, and the washery manager is further prejudiced against the dirt film because it retains moisture and prevents adequate drainage in the hoppers. Were these the only difficulties, they could easily be eliminated by efficient spraying after washing.

The probable reason for the inefficiency of washing in dirty water may be seen by expressing the size of coal and dirt particles in terms of the velocity of water required to cause them to ascend in the upward current of the jig. It has been shown that the velocity required to keep a particle in suspension is :—

$$W = K\sqrt{r(s-1)}.$$

The minimum current for coal and dirt gives the following relations:

$$r_1 = \frac{W^2}{K^2(s_1 - 1)} \text{ and } r_2 = \frac{W^2}{K^2(s_2 - 1)}.$$

If $s_2 - 1$ is replaced by $s_2 - 1 \cdot 1$, W remaining constant, the value or r_2 is increased. Consequently, dirt particles which could not rise in a current of clean water would rise in water with an effective specific gravity greater than unity if the same speed of water current were employed. The washed coal, in these circumstances, would contain more dirt particles than when clean water was used.

It seems desirable to inquire how the advantage of using dirty water suggested by theory could be achieved in practice. There is

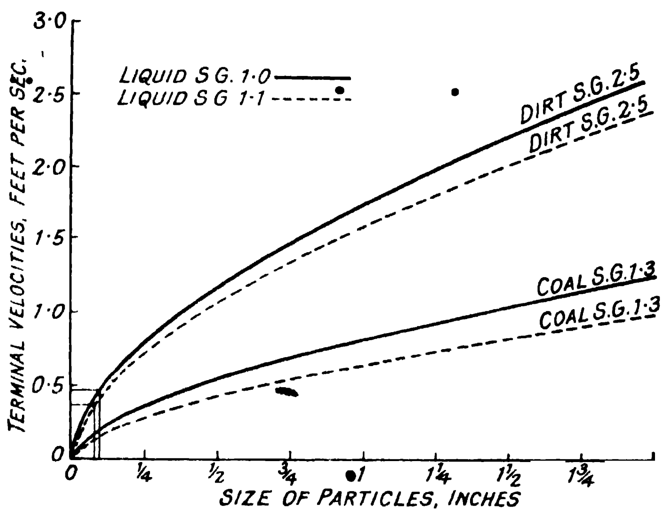


FIG. 6.—Terminal Velocities of Fall. Coal and Dirt Particles in Liquids of S.G. 1·0 and 1·1.

sometimes a shortage of water at a colliery, and frequent changes of the water are expensive. If by a modification of the process, or an adjustment of the washery, good results could be obtained with dirty water, this expense might be reduced considerably.

Fig. 6 shows the terminal velocities of fall of coal particles (S.G. 1·3) and of dirt particles (S.G. 2·5) of different sizes in clean water (S.G. 1) and dirty water (S.G. 1·1). The terminal velocity of fall in clean water has been shown to be $V = K\sqrt{r(s - 1)}$. In dirty water, the equation of motion becomes :—

$$\frac{ar^3s}{g} \cdot \frac{dv}{dt} = ar^3s - ar^3\rho - kbr^2v^2\rho \qquad (24)$$

where ρ is the specific gravity of the water. Solving this equation :—

$$V = K\sqrt{\frac{r}{\rho}(s - \rho)} \qquad (25)$$

The terminal velocities from which the curves in Fig. 6 are constructed are shown in Table 45. In obtaining these values the constant K has been given the values found by de Caux for coal and dirt, these values being more probable values for different sizes than Rittinger's universal constant.

TABLE 45.—TERMINAL VELOCITIES OF FALL OF COAL AND DIRT PARTICLES IN STILL LIQUIDS OF S.G. 1·0 AND S.G. 1·1

Size. in.	Terminal Velocities of Fall (ft. per sec.)			
	Liquid S.G. 1·0.		Liquid S.G. 1·1.	
	Coal.	Dirt.	Coal.	Dirt.
0–1/8	0·236	0·519	0·184	0·479
1/8–1/4	0·356	0·771	0·277	0·711
1/4–3/8	0·463	0·982	0·361	0·905
3/8–1/2	0·546	1·169	0·425	1·076
1/2–5/8	0·619	1·326	0·482	1·222
5/8–3/4	0·688	1·474	0·536	1·358
3/4–1	0·816	1·739	0·636	1·603
1–1¼	0·930	1·972	0·725	1·817
1¼–1½	1·041	2·193	0·811	2·019
1½–2	1·253	2·610	0·976	2·405

It may be seen from Fig 6 that the minimum velocity of current required to bring up 2 in. coal is 1·255 ft. per second (particles being bought up by any current just faster than their own terminal velocities of fall in a still liquid). Suppose that the coal washed were sized before washing between 9/16-in. mesh and 2-in mesh. All the coal would rise. In Fig. 6, a current speed of 1·255 ft. per second cuts the curve for dirt in clean water at the size 9/16 in. No dirt would therefore be brought up with the coal since the dirt below 9/16 in. was removed on the screens.

Suppose now that the same screened coal were washed in water of specific gravity 1·1. All the coal is again brought up by a current of 1·255 ft. per second. In addition, all dirt of size 9/16 in. to 11/16 in. would also rise with the coal, and the cleaned product would be dirtier than if washed in fresh water.

If, however, a current speed of 1 ft. per second were used with water of specific gravity 1·1, all the coal would again rise. But a current of 1 ft. per second cuts the curve for dirt in water of specific gravity 1·1 at about 7/16 in. But since no dirt below 9/16 in. is present, no dirt rises with the coal. There is, in fact, a margin of efficiency in these circumstances. Suppose that a few ½-in. dirt particles were

not removed from the $\frac{9}{16}$-in. to 2-in. coal in screening. In an upward current of clean water of speed 1·255 ft. per second, these particles would rise and contaminate the cleaned product. In an upward current of dirty water of speed 1 ft. per second these particles would not rise and contaminate the clean coal.

The most frequent screening limits in jig washery practice are 0 to $\frac{3}{8}$ in. and over $\frac{3}{8}$ in. (say, $\frac{3}{8}$ to 2 in.). The behaviour of coal and dirt particles of these sizes in clean and dirty water can be illustrated from Fig. 6 by a similar argument. In clean water with an upward current of 1·25 ft. per second, the product from the $\frac{3}{8}$ to 2 in. raw coal would consist of, coal $\frac{3}{8}$ to 2 in. and dirt $\frac{3}{8}$ to $\frac{9}{16}$ in. Using dirty water at the same current speed, the cleaned product would be, coal $\frac{3}{8}$ to 2 in. and dirt $\frac{3}{8}$ to $\frac{11}{16}$ in., a dirtier product than with clean water. But with a current speed of 1 ft. per second and dirty water, the clean product would consist of, coal $\frac{3}{8}$ to 2 in., dirt $\frac{3}{8}$ to $\frac{7}{16}$ in. The product would therefore be cleaner than when clean water was used at the current speed required to recover the largest coal, because of the absence from it of the $\frac{7}{16}$ in. to $\frac{9}{16}$ in. dirt.

Similarly, using 0 to $\frac{3}{8}$ in. raw coal and current speeds of 0·46 and 0·36 ft. per second in clean and dirty water respectively, the cleaned product would consist of: In clean water, coal 0 to $\frac{3}{8}$ in., dirt 0 to $\frac{7}{64}$ in.; or, in dirty water, coal 0 to $\frac{3}{8}$ in., dirt 0 to $\frac{3}{32}$ in. The product from dirty water would contain fewer dirt particles than that from clean water by reason of the absence of the $\frac{3}{32}$ to $\frac{7}{64}$ in. dirt (which may be of considerable proportions).

In order therefore to obtain good cleaning with dirty water it would be necessary to reduce the speed of the upward current (the downstroke of the plunger). The speed of the downward current of water would also be reduced unless the rate of water supply were altered. For reasons which have already been given this would be an advantage. It would seem from these figures that better results might sometimes be obtained by using dirty water and a lower rate of stroke of the plunger. If it were possible to make this adjustment and this were done, the only disadvantages of using dirty water would be the film of dirt on the coal particles. This could be overcome by spraying the washed coal with water to remove the film, and (if necessary) to reduce the salt content of the coal. The water so used could serve as the make-up water for the washery, and the only extra cost would be that of pumping.

The above discussion may seem to suggest that jigs washing $\frac{3}{8}$ to 2-in. coal always leave the $\frac{3}{8}$ to $\frac{9}{16}$-in. dirt in the coal. This is, of course, not the case, for the upward current of the jig only effects part of the separation. Separation according to density differences takes place as the plunger begins to move upward after the downstroke, the water being momentarily still (in a vertical direction). It is probable that the upward current effects most of the separation down to about $\frac{9}{16}$ in., and that the separation of the dirt below $\frac{9}{16}$ in. is more or less confined to the brief intervals between upward and

downward currents. That the intervals are brief does not affect the efficiency of this separation, because the phase in the motion of the particles during which a strict density separation can occur is itself only of very short duration.

The adjustments that would be necessary to compensate for the use of dirty water in an upward current washer would be effected much more easily than in a jig washer, and the examples given serve to illustrate more exactly the variations of the speed of the current that would be required.

The Influence of Friction Between Particles.—It is reasonable to suppose that, although the theoretical treatment outlined indicates with some precision the general principles which govern the classification of particles by upward or alternately upward and downward currents of water, it does not determine with any exactitude the actual accelerations to which the particles are subjected in a washer. Though correct in principle, it is not accurate in degree. A further approximation may be accomplished by the following method of treatment * which takes into account the frictional forces between particles and enables a formula to be deduced for the "acceleration of separation," rather than for the acceleration of individual particles.

Let n and $1 - n$ represent the proportions in which coal and dirt are present in the feed coal, and let μ be the coefficient of friction between coal particles and dirt particles.

The general differential equation of motion will then be

$$\frac{ar^3s}{g}\frac{dv}{dt} = ar^3(s - 1) - kbr^2v^2 \pm n(1 - n)c\mu \quad . \quad . \quad . \quad . \quad . \quad . \quad (26)$$

in which c is a function of the compactness of the material in the jig. The frequency of the contact between coal and dirt is represented by $n(1 - n)$, and the sign of the term $n(1 - n)c\mu$ is "plus" or "minus," according to whether the equation is applied to coal or to dirt. The effect of friction is to increase the rate of fall of a coal particle as it is dragged down by a dirt particle and, at the same time, to reduce the rate of fall of the dirt particle.

The tendency of the mass to separate can be calculated from this general equation. The equation for the movement of the particles will be, for coal,

$$\frac{a_1r_1^3s_1}{g} \cdot \frac{dv_1}{dt} = a_1r_1^3(s_1 - 1) - kb_1r_1^2v_1^2 + n(1 - n)c\mu \quad . \quad . \quad (27)$$

and, for dirt,

$$\frac{a_2r_2^3s_2}{g}\frac{dv_2}{dt} = a_2r_2^3(s_2 - 1) - kb_2r_2^2v_2^2 - n(1 - n)c\mu \quad . \quad . \quad . \quad (28)$$

* Suggested to us by Mr. E. F. Greig of the Safety in Mines Research Board.

The tendency for separation to take place will be determined by the difference in the rates of acceleration—*i.e.*, by

$$\frac{dv_2}{dt} - \frac{dv_1}{dt}.$$

From equations (27) and (28)

$$\frac{1}{g} \cdot \frac{dv_2 - dv_1}{dt} = \frac{1}{s_1} - \frac{1}{s_2} - k\left(\frac{b_1 v_1^2}{a_1 r_1 s_1} - \frac{b_2 v_2^2}{a_2 r_2 s_2}\right)$$
$$- n(1 - n)c\mu\left(\frac{1}{a_2 r_2^3 s_2} + \frac{1}{a_1 r_1^3 s_1}\right).$$

The terms involving v^2 are negligible at the beginning of separation, and the equation reduces to

$$\frac{1}{g} \cdot \frac{dv_2 - dv_1}{dt} = \left(\frac{1}{s_1} - \frac{1}{s_2}\right) - n(1 - n)c\mu\left(\frac{1}{a_2 r_2^3 s_2} + \frac{1}{a_1 r_1^3 s_1}\right). \quad (29)$$

The separation of coal from dirt will be most efficient when $\frac{1}{g} \cdot \frac{dv_2 - dv_1}{dt}$ is greatest, and this occurs when

$$n(1 - n)c\mu\left(\frac{1}{a_2 r_2^3 s_2} + \frac{1}{a_1 r_1^3 s_1}\right)$$

the frictional term, is a minimum.

A number of deductions may be made from equation (29) indicating the conditions under which separation is facilitated or made more difficult. Since separation is facilitated by reducing the value of the frictional term, the influence of various factors may be studied individually :—

(i.) *The Influence of Shape.*—a_1 and a_2 are coefficients to convert the linear dimensions r into the volume of the particle concerned. They will have their greatest values for particles of elongated bodies and their least values for spheres. Separation will therefore be easiest with particles of approximately spherical shape, more difficult with cubical particles, and still more difficult with flat and elongated particles.

(ii.) *The Influence of Size.*—The values of $\frac{1}{r_1^3}, \frac{1}{r_2^3}$ are greater the smaller are the values of r_1 and r_2; so that separation will be easier with large particles than with small particles.

(iii.) *The Influence of Proportion.*—$n(1 - n)$ is a maximum for $n = \frac{1}{2}$. There will therefore be a reduced tendency for separation if coal and dirt are present in nearly equal proportions, and an increased tendency for separation the more the number of particles of one or other constituent of the mixture preponderates.

(iv.) *The Influence of the Nature of the Washing Medium.*—A fluid with lubricating properties will effect a better separation than one without lubricating properties. The presence of a small amount of fine clay in suspension in the washing water is probably an

advantage from this point of view. When gritty particles, such as sand, are suspended in the water (as in the Chance process) or magnetite (as in Conklin process) the frictional forces are increased, and this factor would decrease the "acceleration of separation."

(v.) *The Influence of Compactness.*—The factor c will be greatest when the particles are fed in bulk, relatively close together, or in a condition such that they stick together. It would, therefore, be an advantage, from the point of view of separation, to feed the coal in a stream of water, so that the particles would be wetted and more or less separated from each other before being delivered into the washing chamber. Frictional forces will also be less the deeper the particles are immersed in the washing water or the thinner the bed of material on the screen.

This consideration of the effects of friction between the particles does not invalidate the theoretical conclusions that have been previously reached with respect to the general principles of coal washing. By comparison of the two general equations (1) and (26), it may be seen that the latter is the same equation as the former with an additional term $\pm n(1 - n)c\mu$ added to the right-hand side. For average daily operation, the value of this term may be regarded as a constant. The proportions of coal and dirt (*i.e.*, n and $1 - n$) have a regular and mean value, the compactness of the feed (c) and the coefficient of friction between the particles also have a mean value. Except for special purposes, it is satisfactory to group these factors under one term which, on any one plant dealing with any one type of raw coal, has a constant value.

If the equations already derived were applied to the motion of the particles in the bed of a jig washer, and were revised to take into account the influence of friction, a term would be introduced which would include some function of the compactness of the bed, the proportions of the coal and shale in it, and some mean coefficient of friction. Supposing that the term were represented by λ, the equation defining the speed of upward current required to cause any given particle to rise from the bed would be of the form

$$W = K\sqrt{r[s(1 \pm \lambda) - 1]}$$

the expression $s(1 \pm \lambda) - 1$ replacing the expression $s - 1$ in the earlier equations.

The introduction of the influence of friction suggests that the limiting size ratio for separation of particles by an upward current is lower than given by the relationship

$$\frac{r_1}{r_2} = \frac{s_2 - 1}{s_1 - 1}$$

for the revised relationship would take the form

$$\frac{r_1}{r_2} = \frac{s_2(1 - \lambda) - 1}{s_1(1 + \lambda) - 1} \quad \ldots\ldots\ldots\ldots \quad (30)$$

For coal and shale of specific gravities 1·3 and 2·5 respectively,

the ratio of sizes becomes $5 - 9\lambda$ to 1, which, for all positive values of λ is less than the original ratio of 5 to 1. Until research has been undertaken to determine whether λ is of the order 10^{-2}, 10^{-3}, or even 10^{-4}, the precise influence of frictional resistance can hardly be estimated.

It may, however, be responsible for the fact that the separation of coal from dirt is often more difficult than is expected. Suppose, for example, that λ were to be equal to 0·4. Then for "pure" coal and "impure" coal particles of specific gravity 1·3 and 1·5 respectively and each of size r_1,

$$V_1 = K\sqrt{r_1[1{\cdot}3\,(1 + 0{\cdot}4) - 1]} = K\sqrt{0{\cdot}82\,r_1}$$
$$V_2 = K\sqrt{r_1[1{\cdot}5\,(1 - 0{\cdot}4) - 1]} = K\sqrt{0 - 0{\cdot}1r_1}$$

The value of V_2 is an imaginary quantity, and it would therefore be impossible to effect a separation of the coal and dirt particles by any method depending exclusively for its action upon their terminal velocities of fall in water.

EXPERIMENTAL INVESTIGATIONS OF THE FALL OF PARTICLES IN LIQUIDS

The experiments of Richards have already been referred to briefly. Richards' classical researches on the fall of particles of galena and quartz in water showed that the transition from the Law of Eddying Resistance ($V = K\sqrt{r(s - 1)}$) to that of Viscous Resistance ($V = K_1(s - 1)r^2$), though well defined, was essentially a mergence of one into the other. Each is a logarithmic law, and, for a given material, they may be stated :—

$$\text{Log } V = \log R + \tfrac{1}{2} \log r$$

and

$$\text{Log } V = \log R_1 + 2 \log r$$

where R and R_1 are constants including the terms K, K_1, and $(s - 1)$.

When the logarithms of the size of particles of quartz or of galena are plotted against the logarithms of their terminal velocities of fall in water, the curves are continuous, but show a change of inclination at the critical values. For quartz and galena Richards found these to be as follows :—

	Critical Velocity. mm. per sec.	Critical Diameter mm.
Quartz	28	0·20
Galena	63	0·13

In the Law of Viscous Resistance, as stated, the index of the diameter of the particles is ½. Richards found, however, that the formulæ could be written more accurately :—

$$\text{Log } V = \log 89 + 0{\cdot}67 \log r \text{ for quartz,}$$

and

$$\text{Log } V = \log 240 + 0{\cdot}75 \log r \text{ for galena,}$$

in each case the term $s - 1$ being included in the constants 89 and 240.

Similar researches of a more recent date have confirmed Richards' conclusions and data, and others have shown that, for the Law of Eddying Resistance, the index of r is seldom 2, but is generally between 1·75 and 2.

The fall of particles in water has been examined photographically by Gooskov (*Bull. Soc. de l'Ind. Min.*, 1910, 12, 163; and *Fuel*, 1926, 5, 340), and by Schultz (Thesis for the Degree of Doktor-Ingenieur of the Technische Hochschule, Dresden, and the Bergakademie, Freiberg, 1914).

Gooskov and Schultz both used cinematographic methods to determine the terminal velocities of falling particles. Schultz used single particles, but Gooskov used aggregates (50 gm.) of coal and shale particles. In a mixture of coal and shale particles of equal sizes, the shale particles were always found to fall more rapidly than the coal particles. The average velocities in falling through Gooskov's vessel and the terminal velocities, are shown in Table 46. In general the particles fell less than 0·2 metre before acquiring their terminal velocity, the requisite distance of fall being less for the larger particles than for the smaller

TABLE 46.—AVERAGE AND TERMINAL VELOCITIES OF FALL MM. PER SEC.

Size (mm.)	Average Velocity. Coal.	Average Velocity. Shale.	Terminal Velocity. Shale.
1·6 to 2·5 . . .	60	100	133
2·5 ,, 5·0 . . .	83	150	200
5·0 ,, 7·5 . . .	110	180	220
7·5 ,, 10·0 . . .	107	137	157
10·0 ,, 12·5 . . .	128	170	180
12·5 ,, 15·0 . . .	145	234	305

Schultz used rough and smooth particles of various shapes and consisting severally of coal, marble, zinc blende, iron pyrites and galena. He made extensive experiments confirming Rittinger's theory, but indicating that the values of Rittinger's constants required revision.

THE HENRY THEORY OF COAL-WASHING

The Henry theory of coal washing (*Rev. Univ. des Mines*, 6, 246) is based on laboratory tests performed by M. Henry, who,

instead of examining the fall of particles in water, investigated the classification of particles in an upward current. The research, therefore, started with conditions comparable to those used in practice, and was empirical rather than fundamental.

Henry's first apparatus is shown in Figs. 7 and 8. It consisted of a U-tube, *C*, containing a cock, *R*. To one arm was fitted a device for supplying water at a constant head of pressure, and the other arm, containing a vertical tube *t*, was fitted with a spout *B*. The cock *R* was calibrated to give known velocities of water at a given head. Particles of coal and shale were placed in the U-tube, and the water current caused some of them to overflow from the spout, *B*, into a funnel, *E*, from which samples could be trapped and collected.

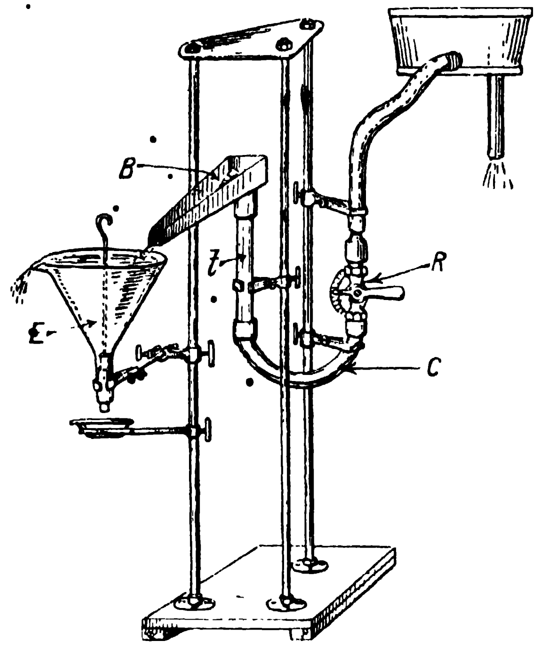


FIG. 7.—Henry Apparatus: General Assembly.

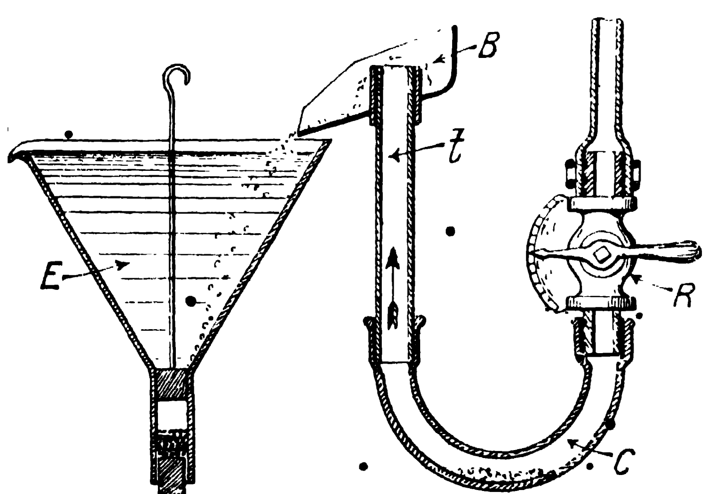


FIG. 8.—Henry Apparatus: Overflow Pipe.

With an "average" sample of coal, Henry found that close screening was required in order to achieve any separation of coal and shale. Postulating conditions that :—

(*a*) All particles with an ash content greater than 50 per cent. must pass without fail as shale,

(*b*) All particles with an ash content less than 10 per cent. must pass without fail as coal,

he found that for coal between 0·5 mm. and 10·0 mm. in size, it would be necessary to screen between the following limits :—

0·5 to 0·8 mm.	2·5 to 4·0 mm.
0·8 ,, 1·5 ,,	4·0 ,, 6·2 ,,
1·5 ,, 2·5 ,,	6·2 ,, 10·0 ,,

In order, therefore, to wash all sizes from $\frac{1}{50}$ in. to about $\frac{13}{32}$ in. with a high efficiency, it would be necessary to screen his coal into six sizes, with an average screening ratio of less than 2 to 1.

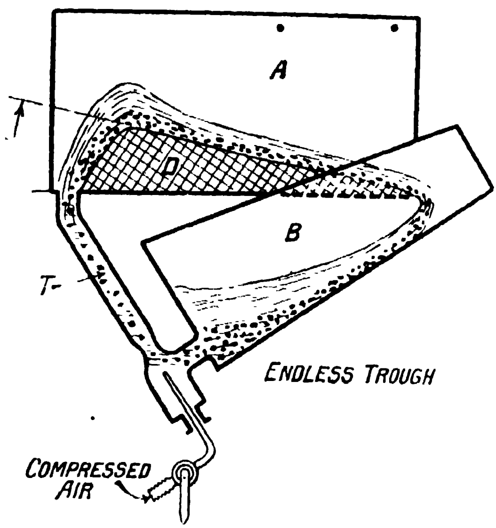


FIG. 9.—Henry Apparatus: Endless Trough.

To a large extent, Henry's conditions may be looked upon as "hindered ascension,' as opposed to "hindered settling." There is little doubt that if it were possible to assume any rigid definitions at all, the laws of one would be similar to some extent to the laws governing the other, the principal difference being that, in hindered settling, the simplest case would be that of a mass of particles falling in a still liquid, in hindered ascension, a mass of particles maintained stationary by an upward current of the liquid, and it is possible that the solution of many of the problems of coal-washing could be solved by a careful and fundamental experimental investigation on lines similar to Henry's.

These experiments convinced Henry that some other factor was called into play in jig washing than the simple buoyant effect of a water current, and he continued his experiments, using a process of alluviation. The apparatus shown in Fig. 9 was employed. It consisted of a sump, *B*, a vessel, *A*, and a pipe, *T*, forming a closed circuit. Coal and water introduced into *B* were circulated by compressed air, and it was found that a deposit, *D*, was formed in the

upper vessel, A. This deposit, D, took the form of a wedge, the longer side having a definite angle of inclination, α. When materials other than coal were used, similar conditions arose, but the angle α had a different value, Henry considering the angle to be characteristic of the density of the material used. Thus, for coal, α had a value of $3° 7'$; for shale, $6° 45'$; for sand, $13°$; and for zinc blende $23°$. The angle of inclination was independent of the size and quantity of material introduced into the circuit or the velocity of the water.

Henry, therefore, put forward a formula $\tan \alpha = k(s - 1)$, k being a constant and s the specific gravity of the material. For coal, shale, sand and zinc blende, k was found experimentally to equal 0·14.

In support of the formula, Henry adduces theoretical reasoning. He considers the particle to have an average vertical velocity, and an average horizontal velocity, the ratio of the two being the tangent of the characteristic angle of inclination. He calls this a "rudder law," * and states it as follows: In a medium which is discontinuous but homogeneous, such as a crowd of particles, a constant acceleration ϕ_m can communicate to a movable particle only a mean uniform motion given by

$$v_m = \phi_m C_t = \text{constant} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (31)$$

where C_t is a time constant depending upon the nature of the particle and the medium.

He considers that, in a jig coal-washer, the particles are subject to a series of brief accelerations during the upward currents, which result in a uniform vertical motion, defined by

$$v_v = g\left(1 - \frac{1}{s}\right) \times \text{constant} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (32)$$

This is the average vertical velocity. The average horizontal velocity is that imparted to a body of diameter r and specific gravity s by a current of velocity V and specific gravity σ (for water $\sigma = 1$). This acceleration, ϕ_h is defined by

$$\frac{\sigma V^2}{2g} \times \frac{\pi r^2}{4} = \frac{\pi r^3}{6} \times \frac{s}{g} \times \phi_h \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (33)$$

Whence, $$\phi_h = \frac{3}{4} \cdot \frac{\sigma}{s} \cdot \frac{V^2}{r} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (34)$$

From his law, the average horizontal velocity, v_h, is ϕ_h multiplied by a time constant, whence,

$$\tan \alpha = \frac{v_v}{v_h} = \frac{4}{3} g(s - 1) \frac{r}{V^2} \times C_t \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (35)$$

Having found by experiment that, for a particle of any given material,

$$V^2 = cr \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (36)$$

* Loi gouvernaille.

c being a constant, the formula may be written

$$\tan \alpha = \frac{4}{3} g (s - 1) \frac{C_t}{c}$$

Or,

$$\tan \alpha = k(s - 1) \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (37)$$

The theoretical basis of Henry's theory is somewhat obscure, and the means of applying it to the conditions obtaining in the bed of a jig-washer are not obvious. There is a considerable significance to be attached to his experimental data, but whether the correct significance has been attached to them is uncertain. Equation (32), for the average vertical velocity of particles submitted to intermittent upward currents, would presumably be equally applicable to a falling particle. The same velocity, according to Henry's theory, would be attained by a particle falling from rest in a still liquid if its fall were periodically arrested and subsequently continued. If the equation be written :—

$$v_m = g\left(\frac{s - 1}{s}\right) \times \text{constant}$$

it immediately recalls the equation :—

$$\frac{dv}{dt} = g\left(\frac{s - 1}{s}\right)$$

based on Rittinger's theory, for the initial acceleration of a particle falling from rest in a still liquid. If this particle fell for a brief period of time, were arrested and again allowed to fall, its average motion for an appreciable falling distance would be uniform and would be given by the equation :—

$$v = g\left(\frac{s - 1}{s}\right) \times \text{constant},$$

the constant referring to the time occupied in falling and in arrest.

This equation, evolved directly from Rittinger's theory, is identical with that put forward by Henry. It has already been shown that, by means of this equation, Rittinger's theory is capable of explaining the ability of a jig-washer to separate coal and shale of widely different sizes. Henry's equation for horizontal motion would therefore seem unnecessary, and, indeed, to complicate the issue.

There are other facts also which suggest that Henry's theory is merely another method of stating Rittinger's theory. Equation (35), for example, may be written :—

$$V^2 = \frac{4}{3} g (s - 1) \frac{r}{\tan \alpha} \times C_t.$$

Or,

$$V = \sqrt{\frac{4}{3} \frac{g C_t}{\tan \alpha}} \sqrt{r(s - 1)}.$$

If tan α is a constant for any given material, the equation becomes simply Rittinger's equation :—

$$V = K \sqrt{r(s - 1)}.$$

In the words of Richards (*Ore-Dressing*, Vol. III., p. 1423), " Rittinger's K seems to be made up of $f\sqrt{2g}$, where f is a factor due to friction," and tan α is a common expression for the coefficient of friction of a particle on a plane.

CHAPTER V

THE HISTORICAL DEVELOPMENT OF COAL-CLEANING PRACTICE

THE important *rôles* played by metals such as iron, copper, tin, lead, silver and gold in the development of civilisation have caused much attention to be paid to processes for the economical extraction of the metallic elements from their ores. Metalliferous ores are found in nature in association with earthy or sandy material of much lower density, and, as a preliminary to processes of roasting and reduction, methods of enrichment by washing have been devised from very early times.

The earliest methods used for coal washing were adopted from ore-dressing practice, so that it is desirable at the outset to refer, briefly, to ore-dressing methods, and to show how they have been applied to the cleaning of coal.

EARLY METHODS OF ORE CONCENTRATION

Probably the oldest method of ore purification, namely, by alluviation, in buddles, strakes, or troughs, was suggested by the stratification of materials which takes place in a river bed. This phenomenon was utilised in natural or artificial streams to separate gold ores, for example, from the associated sandy material. Dams were used to aid the control of the operation by restricting the movement of the heaviest particles, and allowing the lightest material to overflow. Agricola, in the first known book on ore-dressing practice "De Re Metallica," Basle, 1556 (see English translation by H. C. and L. H. Hoover, London, 1912), records that buddles or troughs were in use at that time, whilst the hand-operated jigging sieve "has recently come into use by miners." The operation of jigging is described in the following words: "The metalliferous material is thrown into it (the sieve) and sifted in a tub full of water. The sieve is shaken up and down, and by this movement all the material below the size of a pea passes through into the tub and the rest remains on the bottom of the sieve." The material remaining on the sieve was also stratified according to density, and was then divided, the lighter valueless material on the top being scraped off, leaving a valuable ore. The material which passed through the sieve was washed a second time, with a layer of gravel or small stones laid on the bottom of the sieve to prevent the sandy material from passing through again. This latter practice foreshadowed the employment of a feldspar bed for the washing of fine coal, which Lührig introduced at a much later date. The introduction of the

jigging sieve, which was a great improvement in the methods employed for the dressing of ores, was due to William Humphrey, Assay Master of the Mint, Tower of London, in Agricola's day. It

FIG. 10.—Trough Washers used prior to 1556 (Agricola)

is possible that he learned of it from the German miners, in England, with whom he was associated.

Agricola, referring to strake or trough washing, says: "This method of washing was first devised by the miners who treated tin ore, whence it passed on from the works of the tin workers to those

of the silver workers and others; this system is even more reliable than washing in jigging sieves."

A trough washer used prior to 1556 for tin ores is illustrated in Fig. 10. A boy threw tin stone mixed with mud into the cross launder, whence it passed to two strakes or troughs, D, in streams of water. Two sitting workmen with one hand turned handles, which operated wooden rakes or scrubbers, K, to stir up the settled material and allow the mud to pass forward. In the other hand each workman held a second rake, L, with which "he ceaselessly stirs up the concentrates or tinstone which have settled in the upper part of the strake." The mud was washed forward to a transverse launder, E, and a settling pit, F.

The frontispiece illustrates early methods used for conveying, screening and jig washing of ores. The story is best told in the words of the Hoover's translation of Agricola's book (p. 290): "At Neusohl, in the Carpathians, there are mines where the veins of copper lie in ridges and peaks of the mountains, and in order to save expense being incurred by a long and difficult transport, along a rough and sometimes very precipitous road, one workman sorts over the dumps which have been thrown out from the mines, and another carries in a wheelbarrow the earth, fine and coarse sand, little stones, broken rock, and even the poorer ore, and overturns the barrow into a long open chute fixed to a steep rock. This chute is held apart by small cleats, and the material slides down a distance of about 150 feet into a short box, whose bottom is made of a thick copper plate, full of holes. This box has two handles by which it is shaken to and fro, and at the top there are two bales made of hazel sticks, in which is fixed the iron hook of a rope hung from the branch of a tree or from a wooden beam which projects from an upright post From time to time a sifter pulls this box and thrusts it violently against the tree or post, by which means the small particles passing through its holes descend down another chute into another short box, in whose bottom there are smaller holes. A second sifter, in like manner, thrusts this box violently against a tree or post, and a second time the smaller particles are received into a third chute, and slide down into a third box, whose bottom has still smaller holes. A third sifter, in like manner, thrusts this box violently against a tree or post, and for the third time the tiny particles fall through the holes upon a table. While the workman is bringing in the barrow, another load which has been sorted from the dump, each sifter withdraws the hooks from his bale and carries away his own box and overturns it, heaping up the broken rock or sand which remains in the bottom of it. As for the tiny particles which have slid down upon the table, the first washer—for there are as many washers as sifters—sweeps them off and in a tub nearly full of water, washes them through a sieve whose holes are smaller than the holes of the third box. When this tub has been filled with the material which has passed through the sieve, he draws

out the plug to let the water run away; then he removes with a shovel that which has settled in the tub and throws it upon the table of a second washer, who washes it in a sieve with smaller holes. The sediment which has this time settled in his tub, he takes out and throws on the table of a third washer, who washes it in a sieve with the smallest holes. The copper concentrates which have settled in the last tub are taken out and smelted; the sediment which each washer has removed with a limp is washed on a canvas strake. The sifters at Altenburg, in the tin mines of the mountains bordering on Bohemia, use such boxes as I have described, hung from wooden beams."

Alluviation and hand-jigging were used in Cornwall, during the sixteenth century, for the enrichment of tin, copper and lead ores, and in Derbyshire and Cumberland for lead ores. They were also

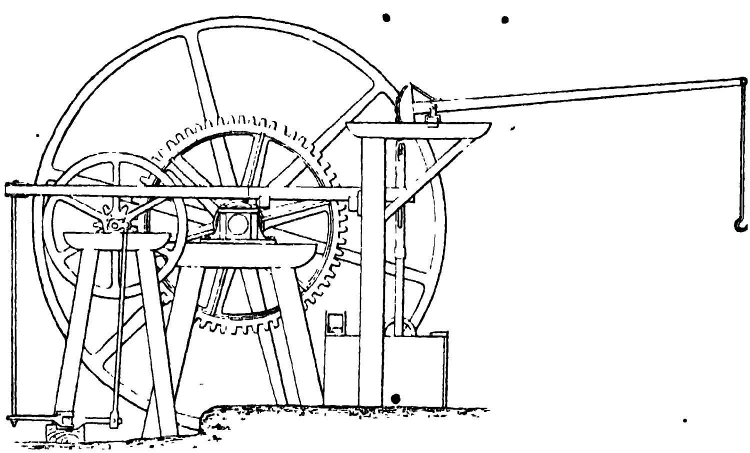


FIG. 11.—Mechanically-operated Movable-Sieve Jig for Ore-Dressing.

practised in the famous argentiferous lead mines of the Hartz Mountains of Hanover, from which district so many mechanical improvements in ore-dressing methods have been introduced.

At a later date the hand-jigged sieve was replaced by a lever and balance weight apparatus by means of which the sieve could be more readily moved upwards and downwards. The next improvement in ore-dressing methods appears to be due to a Cornishman, Petherick, who, about 1830, used a fixed sieve, and obtained the jigging motion in the water by the use of a piston worked by a hand lever (*Quarterly Mining Review*, 1832, see *Proc.* Inst. C.E., 1857–58, 17, 210), and this is the method which is chiefly in use at the present time, except that the motion of the water is produced mechanically.

Fig. 11 illustrates a movable-sieve jigging machine, operated by steam, and used for copper ores in Cornwall (Henderson, " On the

Methods Generally Adopted in Cornwall in Dressing Tin and Copper Ores," *Proc.* Inst. C.E., 1857–58, 17, 195). The method of operation of this machine may easily be followed from the illustration. A layer of iron pyrites was laid on the sieve. The finer material passed through the bed during jigging. Afterwards the upper layer of light gangue was thrown away, the middle layer was rejigged, and the lower layer was suitable for use.

Fig. 12 illustrates a Petherick fixed-sieve—or piston—jig, operated mechanically, which was used at the Par Consols Mine, Cornwall

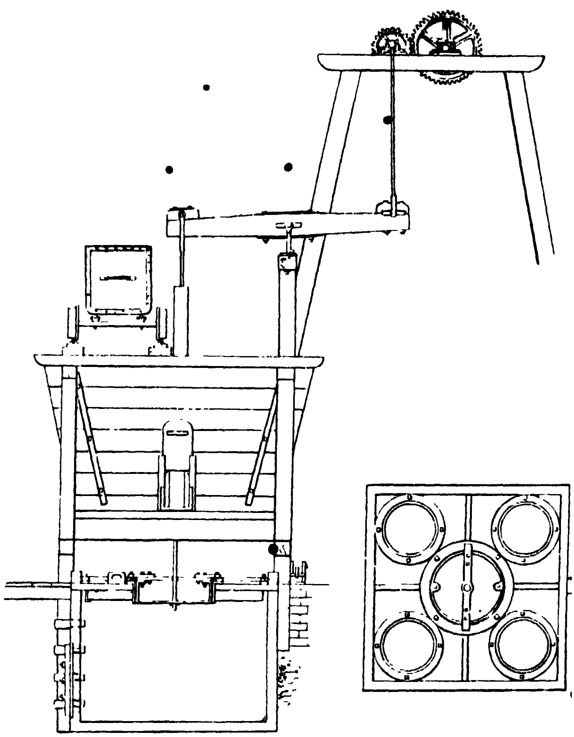


FIG. 12.—Mechanically-operated Fixed Sieve. Petherick Jig for Ore-Dressing.

(Henderson, *loc. cit.*). Four cylindrical chambers were disposed in a wooden tank about a central cylindrical piston chamber, so that, when the piston was depressed, an upward water current passed through the copper ores in the four washing chambers. After jigging for a suitable period, the classified fractions would be recovered in the way previously described.

EARLY METHODS OF COAL CLEANING

The object of processes for dressing metalliferous ores was the enrichment of material which might contain only a few per cent. of

valuable metal, and in which the heavier, and much of the smaller-sized, material might be of great value. The conditions under which coal was obtained were, however, quite different. In the first half of the last century small coal had comparatively little value, and large pieces of coal could readily be freed from adventitious mineral matter by hand picking.

The conditions in England were favourable for this selective mining, for rich thick seams, comparatively free from ash, and yielding relatively strong lumps of coal, were available. On the Continent, however, in certain districts, the conditions were not so favourable, the coal being more friable and the amount of dirt

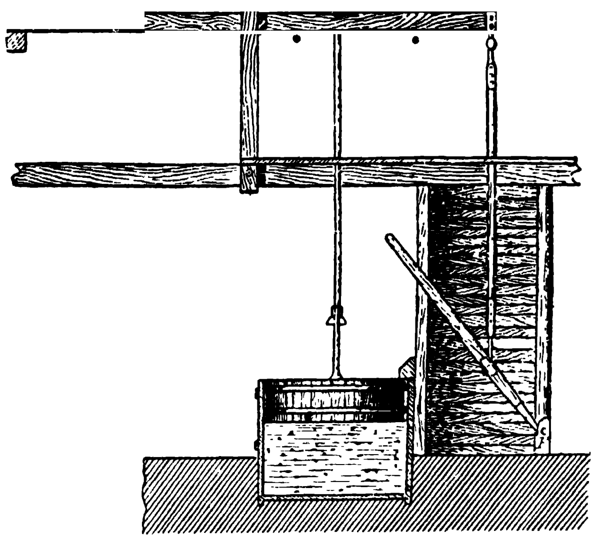


FIG. 13.—Hand-operated Movable-Sieve Coal Jig (1830).

included with the coal was frequently high. It was on the Continent, therefore, that the elimination of dirt from coal by washing first attracted attention. The first appliance to be used was a hand-lever operated movable-sieve jig at Freiburg, Saxony, about 1830 (Gleinitz, etc., "Die Steinkohlen Deutschland und anderer Länder Europas," Munich, 1865). It is possible that this method for coal washing was suggested by a similar method used for the enrichment of ores at Clausthal, in the Hartz district, in the neighbouring State of Hanover.

By the upward and downward movement of the sieve in water, the raw coal on the sieve was stratified according to density differences, the heavier dirt particles settling towards the sieve, whilst the lighter coal particles remained at the top of the bed. When

the separation was thought to be complete, the jigging motion was stopped and the upper layers of clean coal were removed. This type of washer is illustrated in Fig. 13. The movable-sieve washer allowed a rapid and uniform upward movement of water through the coal when the sieve moved downwards, and, by adjusting the balance weight on the lever, the upward movement of the sieve could be retarded.

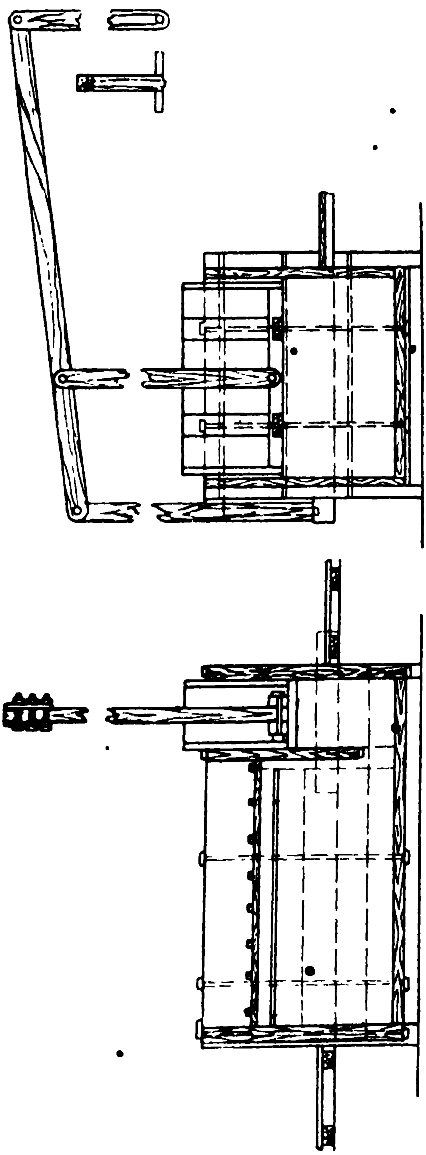


FIG. 14.—Hand-operate ve Coal Jig (1840).

The next stage in the development of coal-cleaning practice was the use of a fixed sieve, the water being made to move through the coal instead of the coal through the water. Although the principle was first applied by an Englishman, Petherick, for ore-dressing, it was first applied to coal cleaning in Saxony, where it was used near Dresden about 1840 (Koettig, Die Steinkohlen des Königreiches Sachsen, Leipzig, 1861). The jigging motion was obtained by the use of a piston, operated by a hand lever. Each washing box was divided into two compartments, so that during the downstroke of the piston in one compartment there was an upward movement of water (and consequently of the coal) in the other section of the box. On the upstroke of the piston the coal settled again on to the sieve.

This type of coal washer was introduced into the Westphalian coalfield at Victoria Stinnes Colliery in 1849, and was adopted at a number of collieries in the same coalfield in 1853 (Sommer, Die Aufbereitung der Steinkohlen, Berlin, 1905). It is illustrated in Fig. 14. Coal of 6 to 70 mm. (¼ to 2¾ in.) size, after

being freed from the coarsest dirt particles by hand picking, was washed in this apparatus, and then re-mixed with the unwashed fine coal and crushed larger coal, the mixture being used for coking purposes.

The method of concentrating by alluviation instead of by jigging was, apparently, first employed for coal washing about 1841, in France and Belgium, where a number of washers of this type were built (*Dinglers Polytech. Journ.*, 1850, 118, 275). An early trough washer at Commentry, France, is illustrated in Fig. 15 in longitudinal section. Several pairs of troughs were connected to a supply trough, *a* (set at right angles to the washing troughs). One such connecting trough, *f*, is shown in Fig. 15. The water flow to each trough was controlled by the gate, *g*. In each washing trough the heavy dirt and the large coal were deposited in the compartment, *b*. The lighter dirt and medium-sized coal were collected in the section, *c*, and the purified coal was obtained in the last section, *d*, from which it was collected in the transverse trough, *e*.

The washing of coal in troughs does not appear to have made

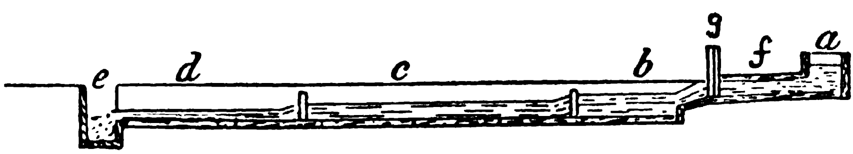


FIG. 15.—Early Form of Trough Washer (1841).

much progress during the early part of the nineteenth century, and the invention of a continuous jig washer by a Frenchman, Bérard, in 1848, acted as a stimulus to coal washing in jigs. The continuous jig washer did not owe its origin to metalliferous ore-dressing practice, as was the case with earlier coal-washing appliances, but was specially designed to meet the needs of coal cleaning. In coal-cleaning practice the product had a low value as compared with the product of ore-dressing, and the feed contained a smaller amount of useless material to be eliminated. In ore-dressing practice the high value of the ore permitted the use of an intermittent process, whilst, in addition, it was often desirable to divide the crude ore into two or three useful products, as well as to remove the valueless sandy material. This division was facilitated by the use of an intermittent process. In coal cleaning, which involved the handling of large amounts of material of relatively low value, the advantages of a continuous process were immediately appreciated. Although Bérard's process was only invented in 1848, and was patented in England in 1849, James Morrison, in 1849, bought cheaply the fine screened slack from the agent of the Earl of Durham and erected a Bérard washer at Newcastle to clean it. He also built beehive coke ovens to use the washed slack, and produced a good quality

coke from material which was previously thought to be worthless The cost of washing was said to be $1\frac{1}{2}d.$ per ton of slack. In 1851 Bérard washers had been built at Loire, Creuzot, Molière, Bessèges

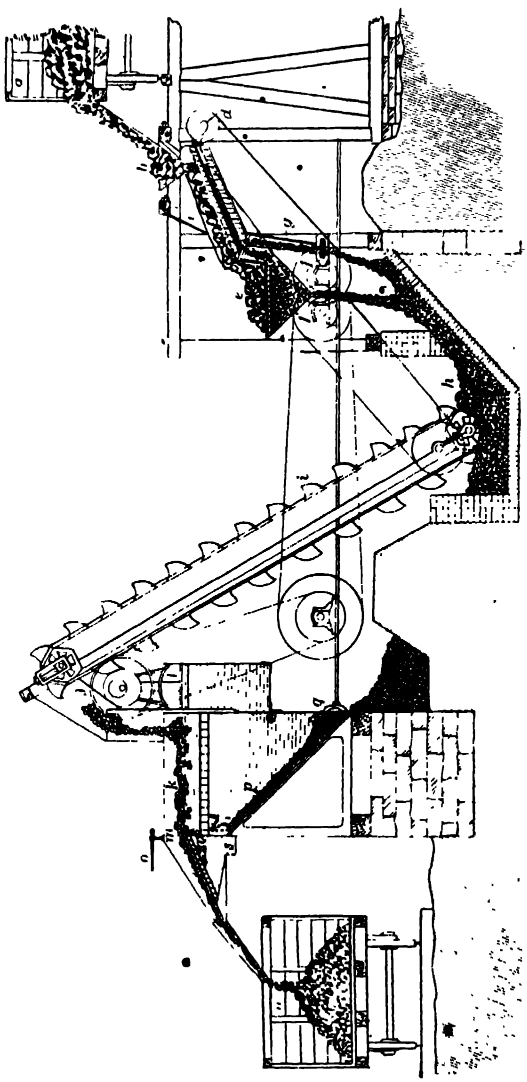


Fig. 16.—Bérard Continuous Jig Washer (1848).

and Spinac, in France, and at Molenbeck St. Jean, Brussels, in Belgium. In 1853 it was introduced into Germany, in Westphalia, and, in 1873, into America.

The main features of the Bérard jig were an elevator, a continuous screening apparatus, a series of wash-boxes, refuse-release valves, and means to remove the coal from the wash-boxes. The

elevator consisted of an endless chain of buckets, passing over top and bottom polygonal drums, driven by shafting which was mounted in suitable pedestals secured to a strong framework. The elevator casing was provided with a gate valve to regulate the amount of coal passing into the buckets.

The sieving apparatus consisted of a rotating cylindrical or truncated conical sieve, or a horizontal rectangular sieve, to which a circular motion was imparted to aid separation. By this means the raw coal was divided into four fractions, each of which was transported by a shoot to a separate wash-box. The four wash-boxes, each of which measured 9 ft. 2 in. by 4 ft. in horizontal section, were made of cast iron and were set adjacent to each other.

Fig. 16 represents a Bérard washer and also illustrates a further feature introduced by Bérard, namely, the crushing of the larger coal before washing. In the plant illustrated, however, the separator or cylindrical sieve was not employed. Coal was unloaded from the wagon, *a*, into a hopper, *b*, from which it passed over a screen, *c*, of 3 in. mesh. The coal over 3 in. mesh rolled down on to a platform, *e*, where it was broken by hand with a maul; the coal less than 3 in. size passed through the screen on to one of ½ in. mesh. The screening was aided by agitation caused by the cam, *d*. The finest coal passed through the ½ in. sieve into the shoot, *g*, and the coal of intermediate size passed through the roll crushers, *f*. The whole of the coal was collected in the pit, *h*. From this point it was taken to the wash-box by the bucket elevator, *i*, and the shoot, *j*. In the wash-box the jigging motion was caused by the movement of the cylindrical piston, *l*. The washed coal flowed over the weir, *m*, over a draining sieve, *r*, into the wagon, *n*. The dirt which settled on to the sieve was removed through a gate valve at one end of the sieve, governed by means of the lever, *o*. The dirt moved down the inclined side of the wash-box, and was removed by operating the sliding door, *q*, at intervals.

The classification of coal into different sizes as a preliminary stage in coal-washing practice was a new and distinct feature of Bérard's washer.

Fig 17 is a view of the first known coal-washing appliance built in England, the Bérard washer introduced by Morrison in 1849 (Birbeck, *Trans.* Chest. and Derby Inst. Eng., 1872, 1, 79). It is stated by Schennen and Jüngst (Aufbereitung, Stuttgart, 1913) that Bérard also introduced a differential mechanism, consisting of a crank and link lever, to make the upward water current faster than the downward current, and so overcome the detrimental effect of "suction" on the upstroke of the piston. He also admitted water below the piston. It would appear that in the Bérard washers built in Great Britain a simple crank was used (as in Fig. 17).

The velocity of the downward currents of water was, however, reduced in Bérard's jig by admitting water through a flap valve underneath the piston during its upward movement. Considerable

variation seems to have been made in the frequency of the strokes of the piston, of which there were from 33 to 150 per minute. The dimensions of the wash-boxes were also varied in a number of plants, doubtless in an attempt to overcome what was found to be the greatest defect of the washer, namely, that the speed of the water current was not uniform over the whole area of the sieve. This defect was, in part, due to the incorrect proportioning of the areas of the sieve and of the piston, and in part to the shape of the lower portion of the wash-box. The direction of travel of the raw coal through the wash-box was from the division plate to the opposite side, that is, at right angles to the direction of travel which is now

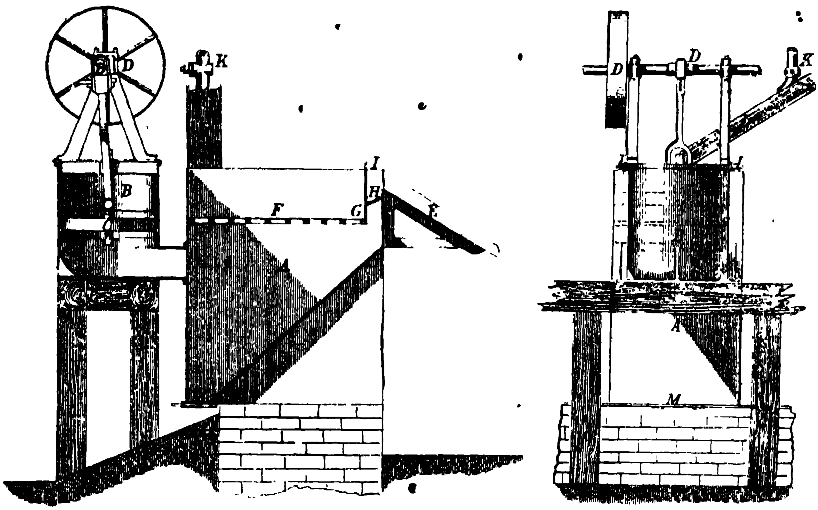


FIG. 17.—Bérard Washer built in England (1849).

adopted in jig washers. As a result the coal was subjected to water currents which were not uniform.

It was claimed that 10 tons of coal could be washed per hour by each washer, at a cost of about 1*d*. per ton. The capital cost was £400, and the water requirements 2,000 gallons of water per day.

After Bérard had demonstrated the commercial possibilities of a continuous process of coal washing, many attempts were made to design other continuous processes on rather different lines. For several years following the introduction of Bérard's washer, a number of jigging machines were devised, but as the principles of classification by jigging were but little understood, many of the machines were cumbersome, and all lacked the simplicity of Bérard's jig, though they incorporated its faults. This state of complexity led the Société de l'Industrie Minérale to appoint a committee, in 1856, to investigate the merits of the numerous jigging machines

which had been devised. The committee concluded that no one type of jigging machine was suitable for all sizes of coal. Some of the jigs were recommended for the washing of nut coal, but only the hand and mechanically-operated intermittent jigs, as well as Bérard's continuous jig, were recommended for fine coal.

In England, the success which attended Morrison's initiative in washing fine Durham coal by Bérard's process drew attention to the great possibilities of coal-washing. Morrison further demonstrated its value by buying up stacks of "smudge" in the Wigan area, and producing a clean coal for coking.

One Bérard washer erected in England for the Wigan Coal and Iron Company had a piston cylinder with a diameter of 3 ft., the strokes being 3½ in. long and at the rate of 150 per minute (*Engineering*, 1868, 5, 242). The raw coal was crushed by rollers to a uniform size and fed to four jigs each of about 8 tons per hour capacity. Power to operate the jigs was developed in a steam engine with a 14 in. diameter cylinder and a 30 in. stroke. About 10 per cent. of material was removed from the raw coal.

Another Bérard jig, erected at Ebbw Vale in 1865 by Messrs. Hawks, Crawshay & Co., of Gateshead-on-Tyne, had four boxes, each of 5 ft. 3 in. length and 3 ft. 3½ in. width (*Engineering*, 1870, 9, 123). The fixed screen had $\frac{1}{16}$ in. apertures, ⅛ in. between centres, and was made of copper, supported on a cast-iron frame. The piston cylinders of the wash-boxes were 3 ft. in diameter. The piston was actuated by an eccentric, and 120 strokes, each of 2½ in., were made per minute. The piston cylinder communicated with the washing chamber by an opening 11 in. high and 22 in. wide. Each wash-box had a capacity of 3¾ tons per hour. The dirt slide was raised when 6 in. of shale had collected on the screen. The washed coal overflowed into a pit-tub, in which the coal remained, the water overflowing to a pond where the slurry was settled. 18,800 gallons of water were used per hour.

The Bérard jig did not, however, find much favour in England, although a number of other types of jig or "bash" washers were erected. Thus, in 1866 (Birbeck, *loc. cit.*), the Sheepbridge Iron Company, Derbyshire, built a washer at Dunston Colliery, in which the coal was classified into four sizes by a revolving cylindrical sieve, each size then passing to a separate wash-box. The wash-boxes were constructed of cast iron, and between each pair was a vertical piston with a stroke of 4 in., 84 strokes being made per minute. The portion of the piston cylinder above the piston was connected by a channel to one wash-box, and the lower portion of the cylinder was connected to the second wash-box. Thus, during the downstroke of the piston there was an upward current produced in one wash-box and a downward current in the second box, and *vice versâ*. The washed coal overflowed into wagons, and the dirt was released by means of a valve. The capacity of the washer was 12 tons per hour, the water used was about 800 gallons

per ton, and the cost of washing 1·97*d.* per ton of washed coal (inclusive of labour, stores, power and repairs).

Differentiation of the length and frequency of the strokes for the various sizes of coal does not appear to have been made. The piston had a diameter of 26 in., and each washing compartment measured 6 ft. by 3 ft. 4 in. The connection between the piston chamber and the washing compartment was only a narrow passage, and for this reason, and because of the great difference in area between the sieve and the piston, the water currents in the wash-boxes were probably not uniform over the whole sieve.

One interesting feature of this washer was the device used to prevent the raw coal elevators from becoming choked. The elevators were fitted with levers and a rocking shaft, worked by a crank on the lower tumbler shaft of the elevator. This mechanism governed a slide over the entrance from the underground hopper into the elevators. By this means the slide was only opened at definite intervals, and the amount of opening could be adjusted to suit the depth of coal in the hopper, so that the buckets could be filled uniformly to a constant depth without fear of overloading.

In another washing plant at Clay Cross, Derbyshire, two jig washers were in use, each of 10 tons per hour capacity, washing coal which had passed through bars spaced ⅜ in. apart (G. Howe, *Trans.* Chest. and Derby Inst. Eng., 1873–74, 2, 86). A horizontally acting piston was used to impart pulsations to the water at the rate of 120 per minute. The dirt was removed from the bottom of the wash-boxes directly into wagons, and the washed coal overflowed into wagons. One feature of this washer was that the slurry was settled in ponds, and the water was re-used.

A somewhat similar type of washer, with a capacity of 10 tons per hour, was in use at Wishaw, Scotland, at a later date (1883). Two wash-boxes adjacent to each other were connected by ports to each side of a piston working in a horizontal cylinder. The piston made 80 strokes per minute, of length 4½ in. The motion of the piston was obtained by means of parallel eccentric motion from a shaft revolving at right angles to the axis of the piston cylinder.

In Germany, a jig designed by Sievers, and introduced before 1870, became fairly popular. In the Siever's jig the pulsations in the wash-boxes were produced by a plunger actuated by an eccentric. The design of his wash-boxes suffered, however, from the same fault as was present in the Bérard jig, namely, the inability to produce a uniform speed of upward current over the whole area of the sieve.

Although the Bérard jig was found to be more efficient than many other jigs, and had the further advantage of continuous operation, it was not universally adopted. Thus, in 1867 (Marsaut, *Bull.* Soc. de l'Ind. Min., 1878), twenty years after Bérard's jig was first introduced, the most satisfactory washer at the works of the Bessèges Iron Company was found to be a mechanically-operated piston jig, which was intermittent in action. The coal was admitted to the

sieve from a hopper until the bed on the sieve was about 12 in. deep. The piston motion was started and the coal was jigged for 200 to 225 strokes of the piston, each stroke being 1¼ to 1½ in. long. The motion of the piston was then stopped and the top layers of coal were scraped off with a shovel. Another charge was admitted to the wash-box and the jigging motion recommenced. After six or seven charges had been treated in this manner, the dirt was removed from the sieve with shovels. This method gave a throughput of about 1½ tons of coal per hour, and the washed coal contained about 8 per cent. of ash, or about 10 per cent. of sinkings in a liquid of S.G. 1·40. The dirt contained about 10 to 12 per cent. of free coal, and the slimes, which amounted to about 12 to 15 per cent. of the total, contained about 25 per cent. of coal. These slimes contained too much dirt for use and were thrown to waste, no method being available for their purification. The effect of the number of strokes on the purification effected by this intermittent type of washer, and the proportion of slimes produced, is shown in Table 47.

TABLE 47.—EFFECT OF THE NUMBER OF STROKES PER CHARGE OF COAL

Number of strokes . . .	50	100	150	200	250	300
Ash per cent. in washed coal .	14·67	11·32	8·32	7·97	7·54	7·32
Slimes produced, per cent. on unwashed coal . . .	5·56	6·50	9·92	10·67	14·87	16·32

The raw coal was the material passing through a ¾ in. square hole screen. It will be observed that a greater proportion of slimes was produced by increasing the number of strokes, owing to disintegration of the coal. Trials made with a Bérard washer gave a throughput of 5 tons per hour, but the washed coal contained about 2 per cent. more ash than the product obtained in the intermittent jig when it had received 225 strokes. It was not until the amount of coal was reduced to 2·5 tons per hour that as good results were obtained with the Bérard washer as with the intermittent jig. It may be remarked that the design of the wash-box of this intermittent jig was an improvement on that of many of the earlier types of jigs because there was no throttling of the water currents between the two compartments.

Marsaut later devised a washer on the lines of the old movable-sieve jig. It consisted of a cage suspended from the piston of a hydraulic cylinder, the cage having a perforated bottom and containing coal to a depth of about 4 ft. The cage was caused to descend by a succession of short, sharp drops through still water, in a rectangular tank about 6 ft. long, 10 ft. wide, and 24 ft. deep. The capacity of the washer was about 15 tons per hour with a power requirement of 1 h.p. One result recorded for the washing of nut

coal of $\frac{3}{8}$ to $1\frac{1}{2}$ in. size in this washer showed that 13·3 per cent. of ash was left in the washed product.

In the early sixties of the last century, Rittinger, an Austrian mining engineer, was studying the theoretical aspects of jigging, and in 1867 he published his classical work on ore-dressing. He derived laws of fall of particles in still water and applied them to jigging practice. From a consideration of the terminal velocities of fall of light and heavy particles of varying size, he concluded that no light particles of S.G. 1·3, and heavy particles of S.G. 1·9, for example, could be separated, if the ratios of the diameters of the largest and smallest particles were greater than

$$\frac{s_1 - 1}{s_2 - 1} = \frac{1·9 - 1}{1·3 - 1} = \frac{3}{1} \text{ (cf. Chapter III.).}$$

This conclusion of Rittinger's had a great influence on jigging practice, and, though at a later date it was found to be too sweeping, it led to a very general practice of close sizing before washing. Rittinger made use of the differences in the rates of fall of particles, varying in size and density, to classify finely-divided ores. He is often said to have invented the appliance known as "Spitzkasten," which enabled particles to be classified into groups of "equal-falling" particles, but this appliance was in existence before 1858 (Smyth, *Trans.* Inst. C.E., 1857–58, 17, 218).

Whilst the development of efficient jig washers for coal was slow, improvement in the design of jigs for the enrichment of metalliferous ores had progressed more rapidly, and once again the coal-mining industry was indebted to the mineral-mining industry for showing the path along which progress in coal-washing practice was possible. In order to prevent small particles of low density from passing through the sieve, it had become customary to wash metalliferous ores on sieves on which was placed a layer of gravel, or rich ore, several inches thick. The use of this device in movable-sieve washers operated by a hand lever was known as the English method, presumably because of its prevalence in ore-dressing practice in Cornwall, Derbyshire and Cumberland.

The English method was first applied to continuous jigs for fine material at Clausthal, in the Hartz district, Germany, where continuous jig washers for mineral ores were first used, some time in the period 1850 to 1864. The main features of the "Hartz" jig were the use of a wash-box which was symmetrical about the partition between the sieve and the plunger compartments, and the form of the box, which at its lower extremity was pyramidal or rounded. The circular-sectioned piston which Petherick and Bérard had used was replaced by a rectangular plunger which covered the whole area of the plunger compartment (except for a small clearance). It was supported by one or two vertical connecting rods, operated by means of crank levers, or by eccentrics. By using a symmetrical wash-box, with equal areas of plunger and sieve, the

speed of the water currents passing through the material in the bed was made more uniform over the whole area. The artificial bed, composed of coarse portions of the ore, or other suitable material, rested on the sieve. The apertures in the sieves were larger than the size of the material treated, and the small, heavy ore particles were able to make their way through the false bed, and through the sieve over its whole area. Valves for the removal of the heavy material were thereby rendered unnecessary. This method was called "washing through the sieve," in contrast to "washing over the sieve," a practice used when washing larger sizes of material. The practice of "washing over the sieve" was used, for example, in the Bérard coal washer, in which the heavier material (refuse) was removed at only one zone, through the refuse-removal valve, and not over the whole area of the sieve.

In 1864, Lührig, who had been connected with the dressing of mineral ores, went to Silesia and saw the difficulty which was experienced in washing fine coal (through $\frac{3}{8}$ in. mesh). To fulfil this purpose, he attempted to use an appliance similar to that used for washing finely-divided metallic ores, namely, the Hartz jig. He also used Rittinger's spitzkasten for the classification of the fine coal into a series of sizes. Several unsuccessful attempts were made at first, but, in 1867, Lührig erected the first feldspar fine-coal washer, at Gluckhilf Colliery, Lower Silesia (Soldenoff, *Proc.* S. Wales Inst. Eng., 1884–85, 14, 88), in which a false bed of feldspar was placed on the sieve. In 1875, Lührig introduced his fine coal washer into Westphalia. Later, he also modified the design of the fine coal washer to make it suitable for washing larger sizes of coal.

Shortly after the work of Lührig in Silesia, a washer containing some of the features of the Hartz jig was introduced in England. This was the Sheppard washer, built by Messrs. Charles Sheppard and Sons, of Bridgend, Glamorgan. It was first built in 1875. Primarily, the design of the Sheppard washer followed the lines laid down by Bérard, though the coal was not necessarily sized before washing. It followed the Hartz jig, however, in that the communication between the piston and the washing compartments was not restricted, as in many earlier types of washer, and the main portion of the wash-box was symmetrical about its axis.

A section of a jig for unsized coal is shown in Fig. 18, which illustrates these features of design. The jigging motion was imparted to the water by a piston, *e*, worked by a crankshaft.

The design suffers in comparison with that of many later types of wash-box in that the coal was admitted near the partition between the piston and the washing compartments and was removed from the side remote from the partition. In most modern types of wash-box the coal travels in a direction parallel to the partition, and because of this the coal is subjected to more uniform water currents.

In Fig 18, near the side where the washed coal overflows, there is an adjustable refuse gate under which the dirt passed, and settled

to the bottom of the wash-box, whence it was carried by a screw conveyor to a dirt elevator. The washed coal overflowed from the sieve compartment, *a*, on to a fixed sieve, G, through which, with the help of a rotating brush, H, the finest coal fell into the subsidiary compartment of the wash-box, *d*, and was conveyed to an elevator. The nut coal passed over the fixed sieve before being loaded into wagons.

To diminish the amount of water passing through the bed during the upstroke, flap valves, *g*, were fitted on the division plate between the main compartment, *c*, and the subsidiary compartment, *d*, of the washer, and these were opened automatically when the piston

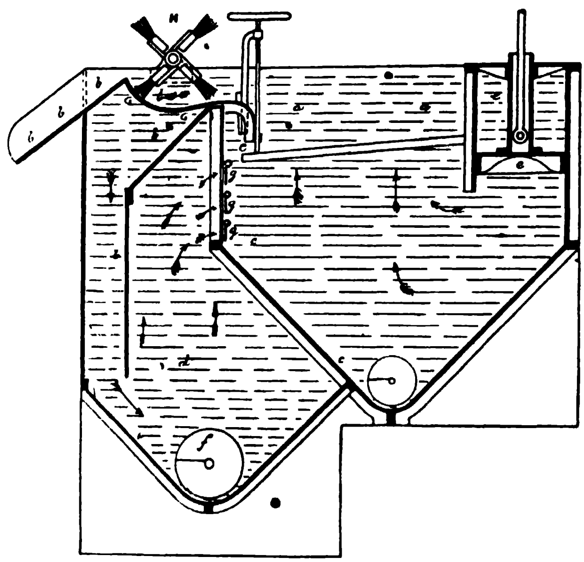


FIG. 18.—The Sheppard Washer (1875).

was raised, allowing water to pass through into the main compartment. This inflow of water compensated in part for the upward displacement of the piston, and so reduced the detrimental effect of the upstroke. In the earliest types of Sheppard washer the stroke was 10 to 20 in., at a frequency of 30 per minute.

The water was removed from the nut coal on a dewatering screen, and much of the water from the fine coal and the dirt elevators drained through perforations in the buckets back to the wash-boxes. There was consequently no external circulation of water with this type of washer, and the subsidiary compartment, *d*, acted to a limited extent as a sludge-settling tank. A plate was fixed in the subsidiary compartment, *d*, of the washer, so that slurry could settle in the sub-section, *b*, undisturbed by the water rising to pass through the flap valves, *g*.

The Sheppard fine-coal washer, introduced in 1885 to treat coal

through ¼ in. mesh, was of the movable-sieve type. Double wash-boxes were employed; in each box the sieve, which consisted of an iron grating with a perforated copper plate, was attached by a

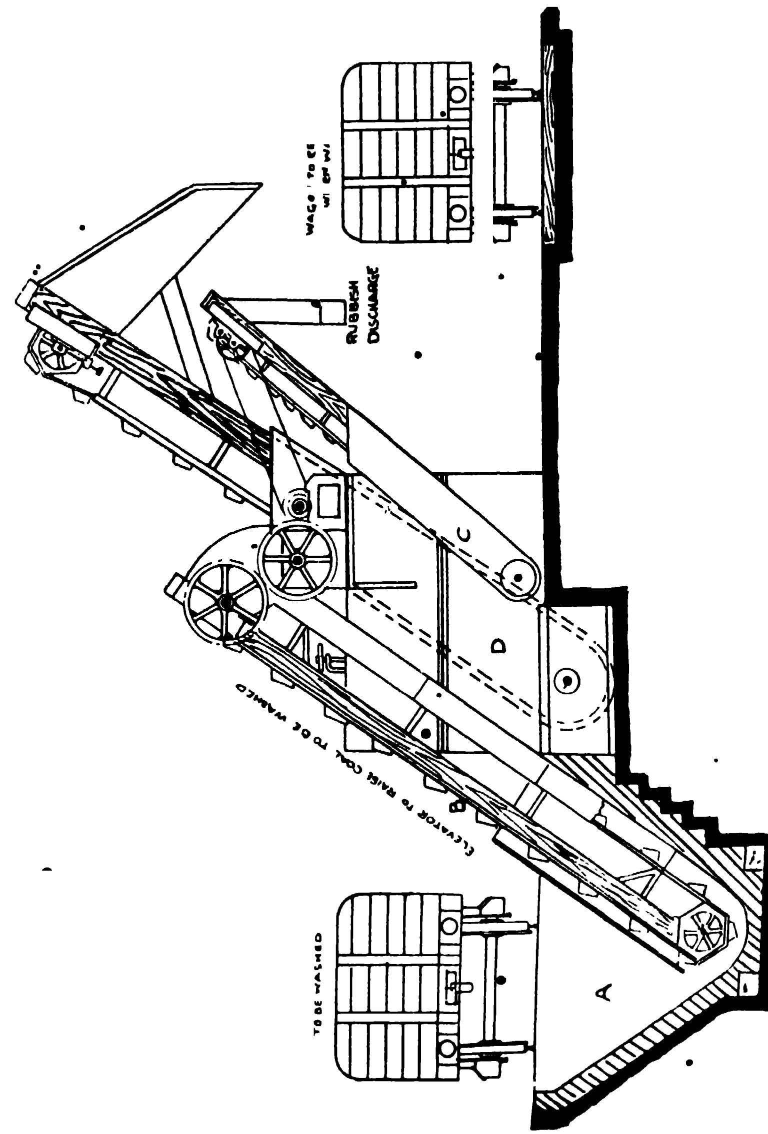
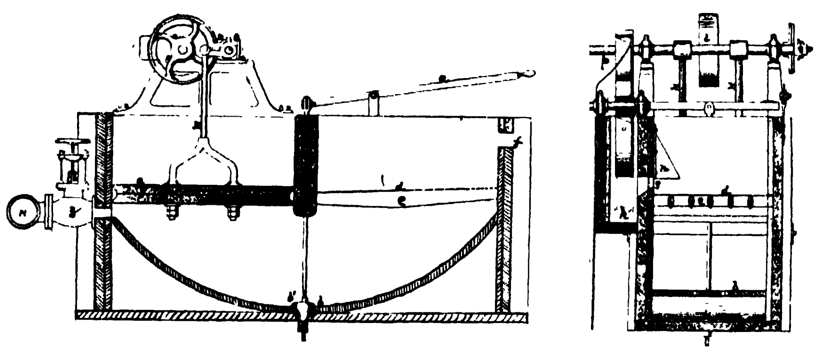


FIG. 19.—Diagram of Sheppard Coal-Washing Plant.

connecting rod to either end of a beam lever. This lever was actuated by a crank so that when one sieve was depressed in one wash-box, the sieve in the other wash-box was raised. About 200 strokes were made per minute. A bed of feldspar was later

used on the sieve. The washed coal flowed over a weir into a compartment outside the main wash-box, where it settled and was removed by means of a screw to an elevator. The external compartment was common to a battery of six or eight fine-coal wash-boxes placed side by side, and was extended beyond the area occupied by the battery of wash-boxes proper so that sufficient space should be allowed for the slimes to settle from the water before its re-use. The clean water was pumped off and was re-admitted to the wash-boxes as required. The settled sludge was removed by a scraper to the washed-coal elevator. An automatic refuse-removal valve was incorporated in the design, a plate being fixed vertically above the sieve so that it projected into the dirt bed. The dirt passed underneath this plate and flowed over a weir into the bottom of the washing compartment, whence it was conveyed to an elevator.

The Sheppard washer proved very popular in Great Britain,



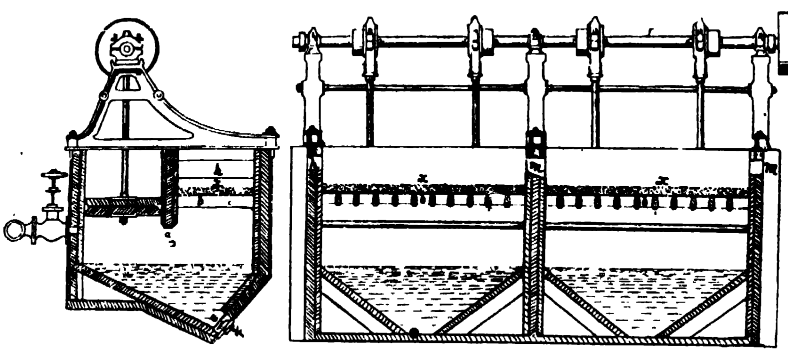
FIGS. 20 AND 21.—Sections through Luhrig Nut-Coal Jig (1879).

particularly in South Wales. There were twenty-four Sheppard plants erected up to 1880, and the number had increased to sixty in 1890, each plant having an average capacity of about 15 tons per hour. Fig. 19 is a diagram showing the general arrangement of a Sheppard washer, and illustrating the simplicity of the earlier types of washer.

D. Cowan (*Trans.* Min. Inst. Scot., 1888–89, 10, 229) records the results of investigations into the efficiency of a number of types of coal washer; for a Sheppard washer at Kinneil, Scotland, the coal contained 15·36 per cent. of free dirt before washing, and 9·06 per cent. after washing, whilst the dirt removed contained 4·25 per cent. of coal. The percentage of dirt removed was thus only 39·9 per cent.

One of the earliest Lührig jigs for washing coal larger than ⅜ in. is illustrated in Figs. 20 and 21 (Rathbone, *Trans.* Inst. Min. Eng., 1879, 29, 159). The wash-box was constructed of wood and was symmetrical about its axis through the partition between the

plunger and sieve compartments. The floor of the box was curved in its cross-section (Fig. 20). The washing compartment was 3 ft. long and 4 ft. wide, and had a perforated plate or sieve, *d*, on which the bed rested. The plunger compartment was made of approximately the same dimensions as the washing compartment, and the plunger, *c*, was rectangular and was fixed to two piston rods, *x*, with forked connections (Fig. 20). Motion was imparted to the plunger by means of a cranked lever from a disc crank which had a dove-tailed groove to receive the crank pin; by this means the eccentricity could be adjusted to vary the length of the stroke of the plunger. The arrangement of the cranked lever reduced the speed of the upstroke, as compared with the downstroke. The plunger made 60 to 75 strokes per minute, according to the size of the material washed; the length of the stroke was also varied from 1¾ to 4 in., being longer for the larger sizes of coal. The refuse-release valve is shown at *n* and *g* (Fig. 21). Lührig's scoop wheel



FIGS. 22 AND 23.—Sections through Lührig Fine-Coal Jig (1879).

(Fig. 21) was fitted to the original Hartz jig to aid the removal of the dirt from the wash-box. The scoop wheel had a number of scoops or buckets, *h*, on its circumference, and was rotated by means of a belt driven from a pulley on the main driving shaft. As the refuse was released from the opening, *g*, it was picked up by one of the scoops and was tipped into a shoot, *p*, for conveyance to the dirt hopper. The use of the scoop wheel allowed the dirt to be inspected and drained before its delivery into the dirt shoot. Finer dirt, which passed through the sieve, was collected in the bottom of the wash-box, whence it was released at intervals by opening the bottom valve, *b*. The washed coal overflowed through the opening, *f* (Fig. 20), and water was admitted through the valve, *z*, underneath the plunger, to reduce further the effect of suction. About 4 to 5 tons of coal could be washed per hour in one wash-box of this type.

The fine-coal wash-box with a feldspar bed, which is illustrated

in Figs. 22 and 23, was similar in type to the one described, although the ratio of the plunger area to the screen area was reduced, and the wash-box was divided into two or three compartments in its length (Fig. 23). A layer of feldspar, *x*, (S.G. 2·6) was placed on the sieve, *b*, to a depth of about 3 or 4 in., the size of the feldspar particles varying from $\frac{3}{4}$ to $1\frac{3}{4}$ in. according to the size of the coal particles washed. The apertures of the screen on which the feldspar rested were larger than the coal treated, but smaller than the feldspar. The plungers made 130 to 150 strokes per minute, of 1 to $1\frac{1}{2}$ in. in length, the length and frequency of the strokes varying with the size of particles treated. The dirt which settled to the bed passed

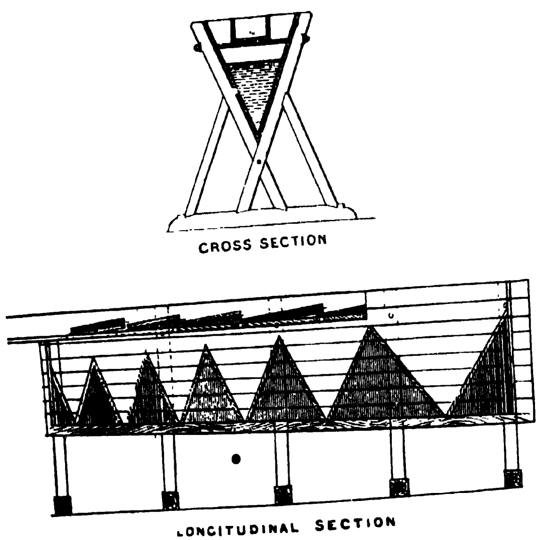


FIG. 24.—Rittinger's Spitzkasten.

between the particles of feldspar, and through the screen, to the bottom of the wash-box, whence it was removed by operating the valve, *k* (Fig. 22). The coal entered the wash-box and passed from one compartment to the next by means of openings, *h*, *m* (Fig. 23). A washer of this type would treat from 3 to $3\frac{1}{2}$ tons of coal per hour.

Great difficulty had previously been experienced in washing fine coal successfully. The defects in earlier jigs which gave rise to this difficulty were twofold, namely, the use of too long a stroke and the varying speeds, in different parts of the bed, of the water currents produced. The use of too long a stroke resulted in fine dirt being carried upwards with the water currents, and in coal being drawn downwards during the upstroke of the piston. The current of water passing upwards through the sieve tended to produce a

number of individual, relatively rapid, upward currents immediately above the holes in the sieve, and to leave a body of relatively quiescent water above the solid parts of the sieve. Methods of correcting this fault by using fine-meshed sieves proved unsatisfactory because of their tendency to clog and so to interfere with the upward water currents. By using a false bed of feldspar, the violence of the pulsations was reduced and the upward water currents were broken up and spread more uniformly over the whole area of the bed. Feldspar particles are parallelopipeds in shape and tend to turn " end on " during an upward current of water, and to rest on their sides during a downward water current, so that in their action they resembled flap valves. Besides distributing the currents more uniformly, they also served to reduce the effect of the suction stroke on the material above the bed.

The spitzkasten used to classify the fines are illustrated in Fig. 24. Their total length was about 20 ft. divided into six boxes, the size of which increased from left to right. The fine coal (through $\frac{3}{8}$ in. mesh) entered the trough above the spitzkasten (on the left-hand side in Fig. 24), and the coal was classified according to size in the various compartments. The settled coal from the base of each compartment was then passed to fine-coal wash-boxes.

A typical Lührig washer, of 50 tons per hour capacity, divided the nut coal (over $\frac{3}{8}$ in. size) into five fractions, namely, 2 to $1\frac{1}{2}$ in., $1\frac{1}{2}$ to 1 in., 1 to $\frac{7}{8}$ in., $\frac{7}{8}$ to $\frac{5}{8}$ in., and $\frac{5}{8}$ to $\frac{3}{8}$ in., each of which was washed in a separate wash-box of the type shown in Figs. 20 and 21. The fine coal (through $\frac{3}{8}$ in.) was classified into five sizes by means of spitzkasten, each fraction being then washed in a feldspar wash-box of the type shown in Figs. 22 and 23. There were, therefore, ten wash-boxes to deal with coal below 2 in. size.

A notable improvement made in the Lührig feldspar washer was to make the coal travel through the wash-box in a direction parallel to the dividing plate between the washing and the plunger compartments, instead of at right angles to it, as had been the practice in the Bérard, Sheppard and other washers. By the alteration of the direction in which the coal travelled through the box, and in the better proportioning of the plunger and sieve areas, and the more favourable dimensions of the wash-box chosen, the Lührig washer constituted a great improvement on previous types of coal washer. Moreover, in washing closely-sized materials, the number and length of strokes could be readily standardised for each material, and washing was therefore facilitated. On the other hand, the number of fractions into which the raw coal was divided before washing greatly increased the initial and upkeep costs of the buildings and plant, and increased the risk of inefficient washing if the supply of raw coal was interrupted.

In Table 48 a number of results of washing fine coal (through $\frac{3}{8}$ in.) are given (G. Blake-Walker, *Trans.* Inst. Min. Eng., 1893–94, 7, 392).

TABLE 48.—RESULTS OF WASHING FINE COAL IN LÜHRIG JIGS (1893)

Capacity of washer, tons per hour	30	30	50	50	50	60	75
Ash per cent. in unwashed coal .	15·8	17·5	11·5	18·0	18·0	22·0	20·8
Ash per cent. in washed coal .	3·8	3·5	1·8	5·4	4·1	4·1	3·4

Such efficient removal of dirt was, however, not obtained without appreciable loss of coal in the refuse. At Dowlais the dirt contained, in one instance, 5·7 per cent. of coal (Cowan, *Trans.* Min. Inst. Scot., 1888–89, 10, 229), and, at Wharncliffe Silkstone, 12 per cent. (Blake-Walker, *loc. cit.*), whilst at Cwm-Avon "the dirt contained appreciable quantities of coal" (*Trans.* Min. Inst. Scot., 1889–90, 11, 147).

The superiority of Lührig's feldspar washer for the cleaning of fine coal was immediately recognised, and the principles of his design were adopted by many continental constructors. The Barop nut-coal washer, for example, was of rectangular cross-section, but with a false bottom sloping at an angle of 30 degrees to the horizontal from the delivery side of the box to the floor at the centre of the box. The ratio of areas of these two compartments was 1 : 1·13. A "knee-lever" differential mechanism (see p. 142) was employed to reduce the effect of suction on the stratification of the bed. The fine-coal washer was of almost the same design as Lührig's, the ratio of the areas of the plunger and the screen being 1 to 1. A "knee-lever" differential mechanism was used to actuate the plunger, and a layer of feldspar was used. Such a washer was erected in 1878 at Zeche Heinrich Gustav Colliery, Westphalia.

The Schüchtermann and Kremer nut-coal washer was of square cross-section, similar to that illustrated in Fig. 14, but was mechanically operated. The design of the fine-coal washer was almost identical with that of Lührig's, an eccentric being used to actuate the plunger.

The Lührig washer was built in large numbers in Germany. In 1880, Lührig became associated with Evence Coppée, who was interested in the erection of coke ovens, for which clean fine coal was required, and the firm of Lührig and Coppée was formed with its headquarters in Mons. This branch had the rights to build the Lührig washer in Belgium, France and the French colonies. In 1888, Lührig became associated in England with Henry Simon, who was also interested in the erection of coke ovens, and a partnership under the style "Simon and Lührig" was formed, the headquarters being in Manchester. This firm erected four or five Lührig washers before 1890, one of the plants being built for Messrs. Merry and Cuninghame, of Glasgow, at North Motherwell Colliery. Mr. Cuninghame then purchased Mr. Simon's rights in the business,

which became " Lührig and Cuninghame " and, later, the " Lührig Coal and Ore Dressing Appliances Ltd." This firm built large numbers of Lührig washers, particularly in Scotland, and since 1917 has been known as " The Coal and Ore Dressing Appliances Ltd."

The washers erected by the Lührig Coal and Ore Dressing Appliances Ltd. were similar to the Lührig washers already described. Those erected by " Lührig and Coppée " were modified in dividing the raw coal into a smaller number of sizes. Coppée usually divided the nut coal (over ⅜ in.) into two sizes only, and the fine coal (through ⅜ in.) into two sizes. These washers became well known as the Coppée nut, bean and pea washers. Feldspar washers for fine coal were built by Coppée in Great Britain until 1922, when the Baum type of jig was substituted.

Although the jig washer was the chief appliance used for washing coal, a number of trough washers were also used. Trough washers appear to have been introduced into England after 1850, and were used in the Wigan area and in Durham to wash the small coal for coke making.

The practice of sizing and cleaning coal did not receive serious attention in Great Britain until about 1880. This was due, in part, to the fact that the seams available were thick and of good mechanical strength, and, in part, to the general practice of loading coal in the mines through a riddle, or with a " fork and pan " arrangement. As a result of this practice much of the small coal, which contained the largest amount of dirt, was left in the mine Monetary fines were also imposed on the miners for sending up dirt with the coal. At many collieries the amount of dirt in the tubs of coal sent to the surface was determined at intervals, and in one pit in Cumberland, for example, an amount of 3 cwt. was deducted from the weight of each tub for every 7 lb. of dirt found in a tub of house coal. At the same colliery, where a cleaner seam of coking coal was also worked, the weight of the whole tub of coal (about 10 cwt.) was deducted if it contained 7 lb. of dirt (*Trans.* Min. Inst. Scot. 1889-90, 11, 181). At some pits in South Wales, where large coal was mined, the miners were required to remove any portion of pyrites or shale from the coal before sending it to the surface. Working in the dim light of a Davy lamp, a miner's output was necessarily restricted if he were required to sort the coal as he loaded it, and the cost of such cleaning in the mine was estimated to be about 6*d*. per ton, compared with a probable cost of 1*d*. per ton if the cleaning had been performed at the surface (Craig, *Proc.* Inst. C.E., 1881, 64, 69).

An increased demand for coal led to the working of thinner seams, and attention was devoted to more efficient methods of preparing the coal. Arrangements which had been satisfactory in 1865, when the coal output was 99,000,000 tons of coal, were not suitable in 1875, when the output had increased, by more than a third, to 133,000,000 tons per annum. Of the increased output,

the export trade, for which the coal required more careful preparation, accounted for 18,000,000 tons in 1875. The pig-iron trade, which had been very small in the first half of the nineteenth century, received an impetus by the introduction of methods of manufacturing cheap steel by Bessemer in 1855 and by Siemens in 1861. The use of these methods of steel manufacture was greatly extended by the introduction of the basic process by Thomas in 1878, and the demand for coke for iron and steel manufacture rapidly increased. With this stimulus to the manufacture of coke, more attention was devoted to the preparation of coal for coking purposes. This resulted in an increased demand for small coal, for it was found that fine coal was more useful for coke manufacture than large coal. Moreover, fine coal was finding an outlet for the manufacture of briquettes in South Wales.

The fixed screen, which had previously been used for the sizing of coal, effected only an inefficient separation and required a large ground space and a steep elevation, often difficult to obtain. A great improvement was provided by the introduction of oscillating or jigging screens which only required a small inclination. The advantages of picking out the dirt by hand from the coal as it travelled forward on a moving belt were also recognised, and these methods of cleaning the coal proved to be much cheaper than methods of cleaning underground. The Coppée and Sheppard washers were built at a number of collieries in South Wales and Monmouthshire in the 'eighties, whilst several were also built in Cumberland and Lancashire. In Durham and Yorkshire trough washers proved the most popular, and an upward-current classifier, the Robinson, was introduced in Durham. The Lührig washer was introduced into Scotland in 1887, and, subsequently, was built there in large numbers. The Lührig washer was also built at a number of collieries in Yorkshire, but it was not until 1894 that the first Lührig washer was built in the Durham coalfield, at Evenwood.

The Robinson washer was devised by R. Robinson, mining engineer for Messrs. Bolckow Vaughan, who owned extensive ironworks, collieries and coke-oven plants in Durham. Mackworth, in 1856, devised an upward-current classifier which was used, but without great success, for coal cleaning and ore dressing. The principle which was employed in the Robinson washer, as also in the Mackworth, was to have a current of water flowing upwards at a velocity such that the lighter coal particles were floated, whilst the heavier dirt particles settled to the bottom of the vessel used. A cone-shaped vessel was employed, with a system of double gates at the bottom, through which the refuse was extracted. The water currents were obtained by means of a pulsometer pump. The Robinson washer, which will be described more fully later, was used at a number of collieries, chiefly in Durham, but also in Yorkshire, Lancashire and Scotland, and mainly where the output was only small. The cost of a washer of 20 tons per hour capacity

was about £200, and the water consumption was estimated to be only about 30 gallons per ton of coal.

A further step in the development of the modern jig washer was made by Fritz Baum, of Herne, Westphalia, in the final decade of the nineteenth century. Baum, who since 1882 had been building coal-cleaning accessories, in 1892 devised a modification of the normal jig washer, in which the pulsations were given to the water by compressed air. He introduced his air jig into England at Middleton Colliery, Leeds, and at Normanton, Yorkshire (*Trans. Inst. Min. Eng.*, 1893–94, 7, 156). Both washeries were of 75 tons per hour capacity, the one at Middleton (which dealt with coal less than 2 in.) having four wash-boxes, and the one at Normanton (in which all coal through $3\frac{1}{4}$ in. was washed) had six wash-boxes, four for nuts and two for fine coal. In the Normanton washer the coal was sized before washing in a revolving cylindrical screen with five concentric drums, the sizes of coal produced being $3\frac{1}{4}$ to 2 in., 2 to $1\frac{3}{16}$ in., $1\frac{3}{16}$ to $\frac{3}{4}$ in., $\frac{3}{4}$ to $\frac{7}{16}$ in. (nuts), and $\frac{7}{16}$ to $\frac{1}{12}$ in. (fines). The material through $\frac{1}{12}$ in. was screened out and added to the fine washed coal without further treatment. The graded coal was carried in a stream of water to the wash-boxes, which were made of steel or cast iron, strengthened by angle irons, and were of semi-elliptical cross-section. Fig. 25 illustrates one of Baum's earliest wash-boxes. Compressed air, at a pressure of $1\frac{1}{2}$ to 2 lb. per square inch, was admitted through the valve, *o*, when the inlet valves were open (and the exit valves closed) and depressed the level of the water in the " air " compartment of the wash-box, *m*. This caused an upward water current through the bed above the sieve, *s*. The opening of the inlet and outlet valves of the air cylinder was governed by the movement of a cylindrical slide working in the air cylinder, and driven by the connecting rod, from an eccentric, *n*. During the downstroke of the slide, air was admitted to the wash-box, and, during the upstroke, the inlet air port was first closed, and then the exit ports uncovered to allow the air to escape. Water was added through the valve, *k*. The unwashed coal entered the washing compartment at the partition between the two compartments of the jig, and overflowed over the weir, *d*. The dirt which collected on the sieve was removed through a double refuse valve, *ff'*, controlled by means of levers, *tt'*. The gate, *f*, was set at a suitable height above the sieve, so that free passage of the dirt underneath this gate was possible. The dirt then passed upwards between the two gates and overflowed over the second gate, *f'*, passing down between the outer and inner casing to the bottom of the wash-box, whence it was conveyed by a screw to an elevator for disposal. About 50 to 70 impulses per minute were imparted to the water for coarse coal, and 75 to 110 per minute for fine coal, the water displacement varying from $2\frac{1}{2}$ in. for the largest coal, to $\frac{1}{2}$ in. for the fine coal, the amount of displacement being regulated by throttling the air valve at *o*.

In the normal plunger type of jig, in which the movement of the water is created by the upward and downward motion of a plunger, driven by eccentrics, the water in the washing compartment accelerates during the first half, and is retarded during the second half of the stroke. After the completion of the downstroke of the plunger, and before the commencement of the upward stroke, there is no appreciable pause. In Baum's jig,

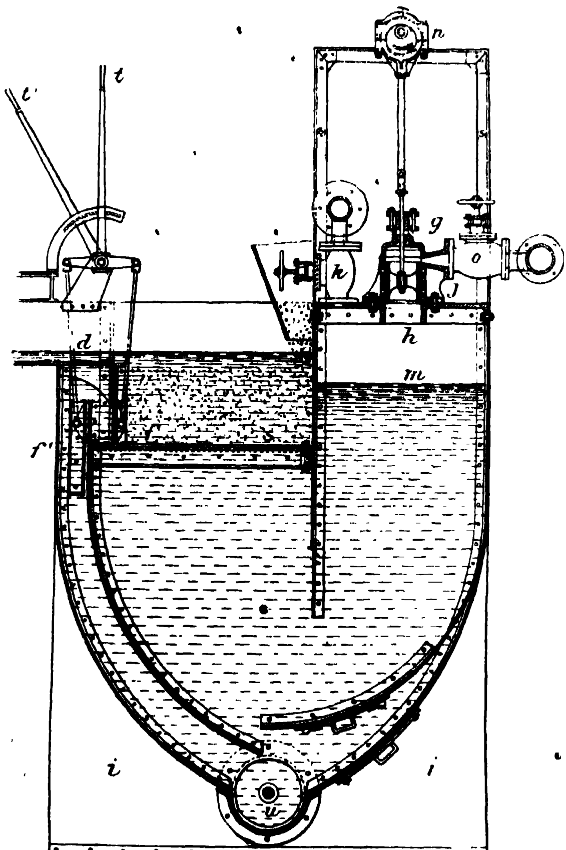


FIG. 25.—Earliest Form of Baum Jig for Sized Coal (1892).

however, the motion of the water accelerated during most of the stroke, and attained its maximum velocity at the end of the stroke. Between the completion of the upward current of water and the commencement of the downward current, the inlet port was closed before the exit port was uncovered, and there was therefore a period during which the water was relatively still. In the initial stages of fall under these conditions, separation according to density differences, independently of size, is possible. In the

early plunger type of jig the downward current of water is caused by a pressure which may be considerable, on account of the production of a partial vacuum during the upstroke of the plunger. In Baum's jig, however, the pressure above the bed on the sieve is only greater than the pressure above the water in the air compartment when the compressed air in this has been released and the pressure has fallen to that of the atmosphere. At this moment the difference in pressure between the two compartments, or, in effect, the pressure exerted on the bed, is only due to the difference in water level between the two compartments. Under these conditions there is never a big pressure exerted on the bed, and there is, therefore, less tendency for it to "set" compactly. In the plunger type of jig this settling of the bed was liable to occur, and during the first part of the succeeding upward water current the bed rose *en masse*, without the individual particles being disturbed. When the compact mass broke up, eddying water currents tended to break through the bed locally, thus remixing instead of separating the materials. In a Baum jig the bed was kept in a less compact condition, and the necessity no longer arose for a feldspar bed to distribute the water currents.

It may be observed from Fig. 25 that Baum used the same direction of flow of coal in the wash-box as Bérard had used, that is, at right angles to the partition between the two compartments of the wash-box. Nevertheless, he was able to wash fine coal after dividing it into only one or two sizes before washing, although he divided the nut coal (from 3¼ to ¾ in.) into four fractions. In 1901, Baum effected a radical change in washing practice, which is expressed in the slogan: "First wash—then classify." He altered the direction of travel of the coal through the wash-box and used a two-compartment jig. In some cases Baum used a second wash-box to re-wash the fine coal and the crushed refuse from the larger sizes of coal.

A more detailed account of the evolution of the Baum washer will be given in a later chapter; it is sufficient here to record that Baum, by departing from the accepted practice of sizing before washing, revolutionised coal washing in jigs.

The practice of close sizing had been accepted by Lührig, Humboldt, Schüchtermann and Kremer, and Coppée, whilst Baum himself in his earlier plants had followed the same practice. Nevertheless, when Baum demonstrated that it was unnecessary, the new practice was followed by other manufacturers, and a Humboldt washery of a later date to wash 80 tons of coal per hour, for example, only divided the coal into two sizes before washing. A modern Baum washer will treat coal up to 3½ in. in size in a single wash-box for capacities up to 50 tons of coal per hour; above this amount, and up to a capacity of 150 tons of coal per hour, a second wash-box is introduced to re-wash the fine coal.

In Europe the Lührig washer made great headway. By the end

of 1889 Lührig had erected 100 coal-washing plants in Saxony (where coal-washing was first practised), Silesia, Westphalia, Prussia, France, Austria-Hungary, and Belgium. Large numbers of Coppée washers were also built in France and Belgium. In the Ruhr-Westphalia district of Germany the number of washers, which was twenty-three in the period 1870–80, had increased to sixty in the next decade, and to ninety-three in the final decade of the last century.

After Bérard's continuous jig washer, the outstanding developments in jig-washing practice are associated with the names of Lührig and Baum. Lührig's feldspar jig enabled fine coal to be cleaned more efficiently than had previously been possible, but to do this he followed Rittinger's theoretical considerations too closely, and needlessly elaborated the design. Coppée built jig washers from Lührig's designs, but reduced the number of sizes into which the raw coal was divided before washing. Baum, however, showed that it was possible to wash unsized coal, in quantities of 50 tons per hour, in one box. Although the use of compressed air to obtain the water pulsations was undoubtedly a contributory factor in enabling this to be done, it was in a larger measure due to the passage of the coal through the jig parallel to the division plate between the sieve and the air compartments. A washer with this type of box was called a "flow" jig (stromsetzmaschine) as compared with a "cross" jig (quersetzmaschine), in which the coal travelled from the division plate across the width of the box.

It was soon recognised that this modification greatly increased the capacity of a wash-box, and the method was adopted in other types of washer, for example, in the Humboldt and the Gröppel. Gröppel, who had taken charge of Lührig's German business in 1893, after the death of the founder, substituted, in 1905, a "flow" jig in place of the "cross" jig which Lührig had built in such large numbers. In a modern Gröppel jig washer—as in the Humboldt—the raw coal is only sized into two fractions before washing, and feldspar is no longer used.

It is therefore apparent that in almost all modern jig washers some of Baum's innovations are embodied, namely, in the use of "flow" jigs, in reducing the amount of sizing before washing to a minimum (if not entirely dispensing with it), or in the use of compressed air to obtain the water pulsations.

In Alabama, America, in 1890, no coal was washed, but the introduction of the Robinson washer, with mechanical improvements made by another Englishman, Ramsay, proved popular, and by 1896 most of the coal used in Alabama for coke manufacture was washed (*Trans.* Inst. Min. Eng., 1896–97, 13, 188).

The Stutz jig was developed in America, but it suffered from the same defects as most of the early jigs, namely, the lack of uniformity of the speed of the water currents over the area of the sieve. A better washer was the Stewart jig, which was a modification of the

earlier movable-sieve washers, in which the jigging motion of the sieve was obtained by means of eccentrics. The sieve, supplied with suitable sides to support the coal, was jigged in a tank of water, and the washed coal passed over the side of the sieve box into a shoot. The dirt was released by means of a slide over a port in one side of the sieve box, so that the dirt fell in the outer water tank. The Stewart jig was adopted at a number of plants in America, and the Lührig jig also proved popular.

In Alabama the floors and roofs of the seams are generally soft and friable, so that much small and impure material is mixed with the coal during mining. Separation of the small dirt by hand-picking was uneconomical, so that resort to coal-washing methods was early found to be necessary. In other coalfields of America, however, the need for coal washing was not so great. In some instances coal for coking purposes was washed to reduce the sulphur content of the coal, rather than for reduction in ash content.

After the beginning of the present century, the increased use of by-product coke ovens in Great Britain led to the erection of washeries of larger capacities than had previously been used. Trough washers, which had been widely adopted in Durham and Yorkshire when the older process of manufacture of coke in beehive ovens was in use, were found to be of too small capacity. They also occupied an excessive ground space, required large quantities of water, and the labour involved proved a big item of expense. At the same time much coal was lost in the refuse. Moreover, the washing water was frequently not re-used, so that besides carrying away a large proportion of the coal as slimes, trouble was often experienced with the river conservators.

Many ingenious attempts were made to improve the efficiency of trough washers. These will be described in a later chapter, but a passing reference might be made to the Elliott trough washer which was devised in 1892, and had a movable series of dams connected to an endless chain so that the deposited dirt was carried to the upper end of the trough, where it fell into wagons. The coal was carried along by the water stream to the lower end of the trough, and was loaded into wagons. About 1913 further improvements were made in trough-washing practice by a Belgian engineer, Antoine France, who developed the Rheolaveur washer, which has had such a marked success since it was first devised.

In 1915, Draper introduced a new upward-current classifier. In recent years various methods of cleaning coal without the use of water have been introduced.

In the thirty years from 1875 to 1905, the annual British coal output increased by about 100,000,000 tons. This increased output necessitated the working of more seams which were thinner and dirtier, so that more careful preparation of the coal was necessary. The use of the riddle underground was discontinued in many districts (W. Scott, *Trans.* Inst. Min. Eng., 1902, 23, 179) and

more fine coal was brought to the surface. By the sizing and cleaning of small coal, the value of slack was much enhanced, and markets were found for material which, although small, was clean and of regular size (G. A. Mitchell, *Trans.* Inst. Min. Eng., 1893–94, 7, 319). During the final decade of the last century the use of electricity for the transmission of power was greatly developed, thus obviating the use of separate boilers and steam engines in a washery.

Another factor which tended towards the increased use of coal washers was the greatly increased output which had to be handled at the colliery surface plant. Where previously a few screens and picking belts had proved satisfactory, a larger output of smaller coal was frequently difficult to clean adequately with existing accommodation at the colliery surface plants. In many instances the increased output would have involved a much greater development of dry-cleaning arrangements (picking belts) requiring much ground space and an army of boys for labour. Since the capacity of a wash-box had been increased from 2½ tons per hour in Bérard's washer to 50 tons per hour in a Baum wash-box, for example, the greater output of a pit could be more efficiently handled by jig washers.

The tendency in Great Britain is to standardise the jig washer along the lines laid down by Baum, that is, to wash the coal before sizing, and to use compressed air as the means to obtain pulsations in the wash-box. Thus, the Coppée washer in recent times has been modified in this manner, and the modern Greaves washer is also of the Baum type. Trough-washing practice with the Rheolaveur washer has proved very successful, and the Draper has recently been preferred among upward-current classifiers. Coal-washing methods depending on selective wetting, such as froth flotation; methods of wet and dry tabling; and methods of dry separation, by the use of spiral separators, by sand flotation, or by pneumatic means, have been devised, but have not, as yet, been widely adopted.

CHAPTER VI

THE PRESENT POSITION OF COAL CLEANING

PRESENT POSITION IN GREAT BRITAIN

ALMOST all the coal mined in Great Britain is screened. The large coal is hand-picked and (in most of the larger coalfields) much of the smaller coal is washed and divided into a number of sizes before marketing. In Durham, however, a large proportion of the gas-coal is sold without any preparation as "run-of-mine," but the coal is sometimes screened and the large coal hand-picked before remixing with the smalls.

Most of the large coal produced in Great Britain is therefore cleaned by hand-picking, but no official figures are available to show the proportion of large to small, or the proportion of the small coal which is cleaned. Since, at the present time, almost all the small coal cleaned is treated by wet processes—or in other words, is washed—an estimate of the proportion of small coal cleaned may be made from a consideration of the capacity and average period of working of the coal washeries erected. We have attempted to do this by collecting data from the washery constructors, and record in Table 49, an analysis of the figures obtained.

TABLE 49.—TOTAL HOURLY CAPACITY OF WASHING PLANTS IN GREAT BRITAIN

Type of Washer.*	No. of Plants.	Total Hourly Capacity (tons).	Average Capacity (tons per hour).
Jigs: Lührig type	314	11,071	35·3
Humboldt type	18	1,352	75·1
Baum type	140	13,787	98·4
Miscellaneous	154	5,025	32·6
Rheolaveur	17	1,295	76·2
Upward-Current and Trough washers	36	2,311	64·2
Totals	679	34,841	51·3

* To avoid the comparison of the number of plants built by different washery constructors, jig washers are classified as Lührig type (including Lührig, Coppée and

The total hourly washing capacities of all types of washer in each coalfield of Great Britain are recorded in Table 50. From these figures an estimate is made of the yearly capacity by assuming that the washeries are working during a 10-hour day for 300 days at two-thirds of the rated hourly capacity.

TABLE 50.—WASHING CAPACITY IN DIFFERENT COALFIELDS IN GREAT BRITAIN

Coalfield.	Total Hourly Capacity of Washers (tons).	A. Estimated Yearly Capacity (tons).	B. Total Coal Production in 1925 (tons).	$\frac{A}{B}$ per cent.
Yorkshire .	9,281	18,562,000	45,273,399	40·8
Derby and Notts.	3,807	7,614,000	32,755,690*	23·3
Northumberland and Durham	3,835	7,670,000	43,448,379	12·8
Staffordshire	1,245	2,490,000	18,700,342†	13·3
Lancashire .	1,813	3,626,000	20,521,439‡	17·7
Cumberland .	424	828,000	4,818,932§	17·2
S. Wales and Mon	8,921	17,842,000	44,629,522	40·8
Scotland .	5,100	10,200,000	33,028,528	30·8
Others .	415	830,000		
Great Britain	34,841	69,682,000	243,176,231	28·3

The figures under the column heading $\frac{A}{B}$ per cent." give a rough guide to the percentage of coal washed in each coalfield and for the whole of Great Britain as an average, but they should only be considered as approximate. The figures suggest that approximately one-quarter of the coal mined in Great Britain is washed.

It will be seen from these figures that coal washing has been most extensively applied in Yorkshire and in South Wales and Monmouthshire. If the figures for Durham and Northumberland are considered separately, the estimated yearly washing capacity expressed as a percentage of the coal produced in 1925 is only 3·5 per cent. for Northumberland. Coal washing is therefore less practised in Northumberland than in any other large coalfield of Great Britain. It is perhaps significant that the average net selling price per ton of Northumberland coal in 1925 (14*s.* 1*d.*) was at least 1*s.* 2*d.* per ton less than the average net selling price of coal

Sheppard washers), Humboldt type, Baum type and Miscellaneous. The miscellaneous jigs include those which use movable sieves, or pistons below the screen plate. Upward current and trough washers include Robinson, Draper, Blackett and Elliott washers, but exclude simple trough washers. All the washers considered have been built since 1875, but most of them during the present century. Compensation for washers now abandoned is probably made by miscellaneous washers not included in the figures. The amount of coal cleaned by dry processes other than hand-picking is at present too small to be included.

* Including Leicester.

† Including Salop, Worcester and Warwick.

‡ Including N. Wales.

§ Including Gloucester, Somerset and Kent.

for all of the other major coalfields (except Scotland), and the highest net selling prices * obtained in those coalfields where the largest proportions of the coal produced were washed. In South Wales and Monmouthshire the average net selling price per ton (excluding anthracite) was 17*s*. 7*d*., and in Yorkshire 16*s*.

Coal washing was first applied to coals used for coking purposes, but more recently it has found considerable favour at collieries producing coals used for steam raising, and for general manufacturing purposes. As an indication of this, the statistics of coal-washing plants built by Messrs. Simon-Carves in Great Britain are recorded in Table 51.

TABLE 51.—STATISTICS OF COAL-WASHING PLANTS BUILT BY MESSRS. SIMON-CARVES IN GREAT BRITAIN

Period.	At Collieries with Coke Ovens.			At Collieries without Coke Ovens.		
	No. of Plants.	Total Hourly Capacity (tons).	Average Hourly Capacity (tons).	No. of Plants.	Total Hourly Capacity (tons).	Average Hourly Capacity (tons).
Before 1921 .	40	3,370	84·2	15	1,595	106·2
After 1921 .	14	1,235	88·2	45	4,462	99·2
Total .	54	4,605	85·3	60	6,057	100·8

From this table it may be seen that the total hourly capacity of Simon-Carves Baum washers built at collieries without coke ovens is much greater than the total hourly capacity of such washers at collieries with coke-oven plants. Coke-oven plants work 24 hours per day, and it is often the practice for washers associated with them to be operated for periods of 16 or even 24 hours. A washer of smaller hourly capacity, but working for a longer daily period, may then be used to deal with the same daily tonnage.

All the coal washed at collieries with coke-oven plants is not used for coking purposes, the proportion probably varying from about 25 per cent. in Yorkshire to over 60 per cent., in many cases, in Durham (where the coal is more friable), with an average of about 30 per cent. for the whole country. The remainder of the washed coal (nuts) is available for gas-making, steam-raising, and other uses.

In 1913 the total coal production in Great Britain was 292·0 million tons, and the total hourly washing capacity 21,625 tons, giving an estimated annual washing capacity of 42·3 million tons, or

* The selling price in Lancashire was exceptional because of the large demand for house coal in a densely populated area.

14·4 per cent. of the 1913 coal production. Since 1914 the hourly washing capacity has been increased by 13,055 tons, giving an estimated additional annual washing capacity of about 10 per cent. of the total coal production in 1925. The average capacity of the pre-war washers (which include large numbers of small Greaves and Sheppard jigs) was only 39·0 tons per hour, compared with an average hourly capacity of 93·4 tons for those erected since 1914.

PRESENT POSITION IN CONTINENTAL EUROPE

Because the statistics available for other European countries cannot be checked so easily as those for Great Britain, the figures recorded in Table 52 may not be so complete. We believe that the figures recorded for France and Belgium are fairly complete, but no allowance is made for washers destroyed during the Great War, nor for the replacement of some of the older washers by newer types. The figures recorded in Table 52 are for the numbers and capacities of jig and Rheolaveur washers in France, Belgium and Germany.* For comparison, the figures for Great Britain, those for Rheolaveur washers in the United States of America, and for various washers in other countries are also recorded.

TABLE 52.—STATISTICS OF DIFFERENT TYPES OF WASHERS BUILT

Country.	Jig Washers.								Rheolaveur.	
	Lührig Type.		Humboldt Type.		Baum Type.		Miscellaneous.			
France	62	3,778	45	2,412	4	250	1	90	52	5,055
Belgium	75	5,508	9	499	2	180	—		47	3,655
Germany	435	29,197	192	19,100	160	15,840	12	2,050	6	870
Great Britain	314	11,071	18	1,352	140	13,787	154	5,025	17	1,295
U.S.A.	—		—		—		—		12	3,055
Other countries	34	2,155	29	1,975	47	4,100	7	1,600	32	2,875
Total	925	52,854	293	25,338	353	34,157	174	8,765	166	16,805

From this table it may be seen that the earliest type of jig, the Lührig type, has been built in the largest numbers, and is still being built to-day. The Baum type is next in order of popularity, but it has been mostly confined to Germany and Great Britain. In Great Britain, the Baum jig has proved more popular than any other type, but in Germany plunger jigs have been built in greater numbers. In recent years the Rheolaveur washer has superseded in favour all types of jig washers in France, and is the only European coal washer which has been greatly developed in the United States of America.

* The washers included are Baum, Humboldt, Lührig, Gröppel, Schüchtermann and Kremer, Meguin, Aufbereitung, Coppée, Rheolaveur and others.

Assuming that the annual yearly capacity of coal washeries may be calculated for other European countries on the same basis as for Great Britain, the figures recorded for hourly washing capacity may be compared with the total coal production, as in Table 53.

TABLE 53.—WASHING CAPACITY IN CERTAIN EUROPEAN COUNTRIES

Country.	Total Hourly Capacity (tons).	A. Estimated Yearly Capacity (tons).	B. Total Coal Production (tons) in 1925.	$\frac{A}{B} \times 100$.
France	11,475	22,950,000	47,046,000	48·8
Belgium	9,842	19,685,000	23,133,000	85·1
Germany	57,327*	114,654,000*	157,000,000*	73·0*
Great Britain	33,296	66,592,000	243,176,231	28·3

France.—In 1923 France produced 40·1 million tons of coal and imported 11·2 million tons from Great Britain, 3·7 million tons from Belgium, and 3·2 million tons from Germany. During and immediately after the War the maintenance of her fuel supplies became an acute problem and much attention was devoted to the improvement of home supplies in quantity and quality. Such progress was made that (with the inclusion of Alsace-Lorraine) 47·0 million tons of coal were produced in 1925, and France also gained a particular interest in the coal production of the Sarre district (whose output was 13·0 million tons in 1925). In 1925 the imports included 9·9 million tons from Great Britain, 5·9 million tons from Germany, and 1·9 million tons from Belgium.

Since 1914 the washing capacity of France has been more than doubled, and considerable increase in the coal-washing capacity of the Sarre district has also taken place. The average hourly capacity of the washers built before the War was about 55 tons per hour, but the new washers built had an average hourly capacity of nearly 100 tons. It is of interest to note that the new washers built were nearly all Rheolaveur washers.

Belgium.—In 1913 Belgium produced 22·8 million tons of coal, and in 1925 slightly more than this quantity. Only 15·3 per cent. of the coal produced in Belgium in 1925 contained over 25 per cent. of volatile matter, most of the coal being very "lean" and, being friable, produced a lot of dust. Belgium cannot produce sufficient coal for her own needs, especially for coke manufacture, and in 1925 imported 3·8 million tons from Great Britain (compared with 5·7 million tons in 1913) and 4·9 million tons from other countries, chiefly Germany. Great care has therefore been devoted to increasing the

* 1913 figures; see p. 130.

value of the home supplies of coal. In 1914 Belgium had an hourly washing capacity of about 3,500 tons, equivalent (on the basis previously used) to about 30 per cent. of the total coal production. Many washeries were destroyed during the War, however, and since then new washeries with an hourly capacity of about 6,000 tons have been built, with an estimated annual capacity of over 50 per cent. of the annual coal production in 1925. The average hourly capacity of the pre-war washeries was 52 tons per hour, compared with an average of 136 tons for those built since the War. It will therefore be appreciated that Belgium is now in possession of some of the most up-to-date and efficient washing plants in Europe. Since 1923 the Belgian Railways Administration have entered into an agreement with the Coal Owners Federation to adopt a sliding scale for coal payments according to ash content. Above a basis of 12 per cent. ash content, a rebate of 2½ per cent. in price is made for each 1 per cent. of ash up to 17 per cent.; from 17 to 20 per cent. ash content, the rebate is 3 per cent. per unit of ash.

Germany.—In 1913 the anthracite and bituminous coal * (*Steinkohle*) production in Germany (including Sarre and Lorraine) was 157·0 million tons and the hourly washing capacity was about 57,000 tons. On the basis previously assumed, the annual coal-washing capacity expressed as a percentage of the total production in 1913 was 73·0 per cent. If the coal production for the eastern part of Upper Silesia be included, the total coal production in Germany in 1913 was 189·9 million tons, and the estimated total annual washing capacity (neglecting small numbers of washers erected in Upper Silesia) expressed as a percentage of the total coal production in 1913 is reduced to 60·3. Within the boundaries of modern Germany the total hourly washing capacity has been increased by 10,830 tons since the War, but the total production was reduced to 132·7 million tons in 1925. Germany has therefore the greatest total hourly washing capacity of any country in the world, and it seems likely that the percentage of the total coal washed is higher than in any other country.

The reason for the extensive practice of coal washing in Germany is that the seams are steeply inclined and are very much faulted (as in France and Belgium), and, moreover, the coals are generally very friable and produce a lot of smalls. For example, at Zeche Westfalen I/II (after the removal of 12·5 per cent. of dirt) from the 80 mm. to 0 ($3\frac{1}{8}$ in. to 0) coal, 17·4 per cent. of large nuts, 80 to 30 mm. ($3\frac{1}{8}$ to $1\frac{3}{16}$ in.), 6·6 per cent. of small nuts, 30 to 11 mm. ($1\frac{3}{16}$ to $\frac{7}{16}$ in.), and 76·0 per cent. of fine coal, 11 mm. to 0 ($\frac{5}{16}$ in. to 0) were produced.† The coking coal (*fett kohle*) is the most friable, and this type of coal represents the largest proportion of the coal mined. For example, in 1925–6 in the Westphalian coalfield (where 80 per cent. of the

* *Braunkohle*, brown coal, is not washed.
† Taschenbuch für Brennstoff Wirtschaft und Feuerungstechnik, Halle, 1928.

total German "stone" coal is produced) 68·7 per cent. was coking (*fett*) coal, 20·3 per cent. was gas (*flamm*) coal, and 11·0 per cent. was lean (*ess* and *mager*) coal.

The use of a large proportion of the coal for coke manufacture has stimulated the development of coal washing in Germany. In 1913 about 40 million tons of coal were used for coking purposes and 6·5 million tons for briquette manufacture, the total amount of coal used for these purposes being 30 per cent. of the total stone coal production (excluding Eastern Upper Silesia).

In Germany the modern practice of coal cleaning is to hand-pick the lump coal over 80 mm. (3⅛ in.) size, and to remove the dust less than about $\frac{1}{50}$ in. before washing. The coal is then divided into three fractions, namely, washed coal, middlings and refuse. Sometimes the large middlings are crushed and rewashed with the small middlings in a rewash box, and the product is used for boiler-firing. Froth-flotation processes, to clean the slurry, are also being adopted at a number of washeries. An interesting development in Germany has been the amalgamation of the two firms who have built the largest numbers of washers in Germany (Schüchtermann & Kremer, and Baum A.G.) with Aufbereitung A.G. (the German agents for the Rheolaveur washer) under the style Schüchtermann & Kremer-Baum A.G. für Aufbereitung.

Other Continental Countries.—Other continental countries which produce considerable quantities of coal are Poland, Russia, Czecho-Slovakia, Holland and Spain. The present position of coal-cleaning in Poland and Czecho-Slovakia is somewhat difficult to assess owing to the fact that the boundaries of these countries are of recent arrangement, and it is difficult to obtain reliable information regarding the number of coal-washing plants available. By including, for Poland, those washers built before the War in Upper Silesia and, for Czecho-Slovakia, the washers built in Austria Hungary, approximate figures may, however, be given.

The coal production within the boundaries of what now constitutes Poland has decreased from 40·7 million tons in 1913 to 29·0 million tons in 1925.* In Czecho-Slovakia the coal production has decreased from 14·3 million tons in 1913 to 12·7 millions tons in 1925, and the production in Austria and Hungary is now negligible. In Russia the coal production in 1913 was 29·0 million tons, but fell to less than one quarter of this total during and immediately after the War period, but it is now increasing, and was 17·6 million tons in 1925. In Holland, where only 1·9 million tons of coal were produced in 1913, the total has gradually increased until it was 6·8 million tons in 1925. In Spain the coal production has increased from 4·0 million tons in 1913 to 5·9 million tons in 1925.

An estimate of the total hourly washing capacity and the percentage of the coal washed is given in Table 54.

* In 1927, about 38 million tons.

TABLE 54.—ESTIMATE OF WASHING CAPACITY IN VARIOUS CONTINENTAL COUNTRIES

Country.	Total Hourly Washing Capacity (tons).	A. Total Annual Washing Capacity (tons).	B. Total Coal Production in 1925 (tons).	$\frac{A}{B} \times 100$.
Poland . . .	1,950	3,900,000	29,080,000	13·4
Russia . . .	2,254	4,508,000	17,637,000	25·5
Czecho-Slovakia .	4,140	8,280,000	13,602,000	60·9
Holland . .	2,087	4,174,000	6,848,000	61·0
Spain . . .	1,173	2,346,000	5,865,000	39·6

It will be seen that the estimated percentage of coal washed is high in both Czecho-Slovakia and in Holland. Most of the washers in Czecho-Slovakia, Poland and Russia were built before the War, but in Holland more than half the total capacity has been added since the War. Most of the Dutch coal is used for coke manufacture, and is of a type similar to Westphalian coking coal, being somewhat friable. One interesting feature about coal-cleaning in Spain is that more froth-flotation plants have been built than in any other country except Germany.

PRESENT POSITION IN JAPAN

In Japan proper and in Manchuria there are a large number of small collieries where the practice of coal washing has been adopted. Most of the coal washers have been erected since the War period. A summary of the numbers and capacities of coal washers is given in Table 55, based on figures recorded by Taiji Iyoku (*Journ. Fuel. Soc. Jap.*, 1927, 6, No. 53), for a translation of which we are indebted to Mr. Tadaji Shimmura, of the Imperial Fuel Research Institute.

The miscellaneous jig washers include twenty-two Kyōēkisha jigs, of 870 tons per hour total capacity, which are of Japanese design. Most of the other washers have been adapted from well-known European designs, five of them (it is interesting to note) originating in Great Britain. The popularity of the Blackett washer is very marked, the total capacity being greater than for the Blackett washers erected in Great Britain. Only one of these washers was supplied by the English firm (Messrs. M. Coulson & Co., Ltd.), who hold the English patent rights. No patents were taken out in Japan, and the design was therefore copied for a large number of other plants. All the washers are of small capacity, much lower than is customary in Europe.

The total coal production in Japan was 21·3 million tons in 1913,

TABLE 55.—NUMBER AND CAPACITY OF COAL WASHERS IN JAPAN.

Type.	No. of Plants.	Total Capacity (tons/hr.).	Average Capacity (tons/hr.).
Miscellaneous jigs . .	72	2,472	34·3
Baum	20	827	41·3
Humboldt	3	115	38·3
Blackett	24	945	39·4
Rheolaveur	2	44	22·0
Robinson	2	25	12·5
Elliott	2	30	15·0
Draper	1	16	16·0
Concentrating tables . .	3	19·5	6·5
Froth flotation (Minerals Separation) . . .	1	30	30·0
Total . . .	130	4,523·5	34·9

but had increased to 29·2 million tons in 1925. In 1913 the hourly washing capacity was only 567 tons, so that the estimated annual washing capacity was only 4 per cent. of the coal production. During the Great War, and particularly since the War, washers with a total hourly capacity of about 4,000 tons have been built, the estimated additional washing capacity being 27 per cent. of the total coal production in 1925.

The new interest which Japan is taking in coal preparation is illustrated by the fact that during 1924–26 a screening and washing plant was erected by Bamag-Meguin A.G. for the South Manchurian Railway Company, at Fushun colliery, with a total capacity of 1,000 tons per hour, this being claimed to be the largest coal preparation plant in the world.*

CHINA.—In China, although 20·5 million tons of coal were produced in 1924, we have only been able to trace the existence of two washeries, so that the proportion of coal washed is probably very small.

PRESENT POSITION IN THE UNITED STATES OF AMERICA

It is difficult to state the present position of coal-cleaning in America, for complete statistics of the washers erected are not available. Many of the washers used are—in European eyes—of rather primitive design, but the need of making efficient recovery

* This plant is not included in the statistics given by Mr. Iyoku.

of saleable coal may not be so great in America as in Europe. In mining practice, for example, by the greater adoption of the pillar and stall method of working, larger quantities of coal are left in the mines, and a larger percentage of coal actually got by the miner is thrown back into the goaf. The average extraction of workable coal in U.S.A. is only 65 per cent., and of the 35 per cent. lost, 19 per cent. was classified as avoidable (" U.S. Coal Commission Report," III., 1841). By making a low percentage extraction it is often possible to send only the best quality of coal to the surface. The total production of coal in 1910 was 492·5 million tons (including 75·4 million tons of anthracite) and 578·2 million tons (including 55·2 million tons of anthracite) in 1925.

In America the bituminous coal seams are usually flat, the roofs and floors are generally good, and the average thickness of the seams worked is over 5 ft. It has therefore been possible to win the coal without much contamination with free dirt. Formerly much of the coal mined was sold as " run-of-mine " without any screening or cleaning but, whilst a large proportion of the output is still sold unscreened, intensive competition has led to the development of screening and hand-picking of the larger sizes for the domestic market. In Alabama only, the soft and friable nature of the roofs, and the presence in the seams of soft dirt partings, has compelled extensive use of coal-washing methods. Official figures are given for the actual tonnage of coal washed.* These are recorded in Table 56.

TABLE 56.—AMOUNT OF COAL WASHED IN U.S.A.

Year.	Amount of Coal Washed (tons)	Clean Coal Production (tons).	Clean Coal per cent of Total Output.
1906	10,425,455	9,251,946	2·7
1913	25,051,801	22,069,691	4·6
1923	22,364,986	20,140,385	3·6

The amounts of coal washed in 1923 in certain coal-producing States are recorded in Table 57.

The amount of dirt washed out was only 7·7 per cent. in Pennsylvania and averaged 10·0 per cent. for the whole country. These figures show how different is the American position from that of Europe, for Alabama is the only large coal-producing State in which a large proportion of the coal mined is washed. Moreover, of the total amount of coal washed in U.S.A., 61 per cent. is Alabama coal. In the biggest coalfields, namely, Pennsylvania, West Virginia,

* Mineral Resources of the U.S., 1924, Part II.

TABLE 57.—AMOUNTS OF COAL WASHED IN VARIOUS STATES IN U.S.A. (1923)

State.	Coal Washed. (tons).	Clean Coal Produced (tons).	Refuse Removed (tons).	Clean Coal (per cent.) of Total State Output.
Alabama	13,570,913	12,285,695	1,285,218	60·1
Pennsylvania	2,705,955	2,496,843	209,112	1·5
Illinois	1,718,971	1,520,819	198,152	1·9
West Virginia	1,258,192	1,144,693	113,499	1·1
Washington	907,339	722,320	185,019	24·7
Tennessee	445,975	390,275	55,700	6·5
Michigan	218,180	197,154	21,026	16·8
Georgia	25,917	24,161	1,756	32·0
Other States	1,513,544	—	—	—
Total	22,364,986	20,140,385	2,224,601	3·6

Illinois, Kentucky and Ohio (which together produce about four-fifths of the bituminous coal), only about 1 per cent. of the coal produced is washed.

In Europe, coal washing was first developed extensively to wash the small coal for coking purposes, and although more coal is used for coking purposes in America than in any other country, the whole of the run-of-mine coal is usually used, and a sufficiently low ash content is possible. Of the total coal charged to coke ovens in 1923 only 16·6 per cent. was washed (9·9 per cent. in Alabama alone), but the washed coal used for coking amounted to 69·7 per cent. of the total washed coal available. Nearly all (96·3 per cent.) of the coal charged to the coke ovens in Alabama, and all the coal coked in Colorado, Georgia, New Mexico and Washington was washed, in 1922. Except for Alabama, however, these States are not large coke producing States.

The Wyoming and Lehigh districts of Pennsylvania yield rather more than two-thirds of the present American anthracite production, and the Schuylkill district of Pennsylvania the remainder. The anthracite seams in Wyoming and Lehigh are flat and fairly clean, but in Schuylkill the seams are steeply inclined, folded, badly faulted, and are of variable thickness. The anthracite produced is, therefore, relatively dirty. The reserves in the Wyoming and Lehigh districts are limited, but it is estimated that the Schuylkill district "contains most of the coal that will be produced in the next seventy-five years" (Dominion Fuel Board No. 5, Ottawa, 1925). The necessity for more careful preparation will, therefore, become more important in the future than it is at present, and the great interest

displayed by the anthracite operators in modern methods of coal-cleaning is perhaps an augury of future development.

In the early days of the anthracite mining industry run-of-mine coal, from which the sizes less than $1\frac{1}{4}$ in. were removed, was sold, the finer sizes being left in the mines. Later, the demand for sized fractions led to the screening of the saleable product and to hand-picking of the larger sizes. Subsequently, the large coal was broken to produce domestic nuts, and the smalls (about 15 to 20 per cent. of the total) resulting from this treatment was dumped on culm-banks, there being no demand for it. As uses were found for the smalls, however, the culm-banks were washed, this movement being stimulated about 1915 by the scarcity and high prices obtaining.

In 1890, 76·9 per cent. of the anthracite shipped was larger than pea size, 23·1 per cent. being pea or smaller. In 1912, however, the percentage of domestic sizes had fallen to 60·8 per cent., and that of pea and steam sizes had risen to 39·2 per cent. During the next ten years the demand for peas decreased from about 11·5 to about 8 per cent. of the total, and the figures for 1923 were: Domestic sizes, 63·7 per cent.; peas, 8·1 per cent.; buckwheats, 28·2 per cent.

The introduction of the Chance process in 1921 enabled the larger sizes to be cleaned more easily than had been possible or customary in jigs, all sizes up to 4 in. being cleaned, and the Chance process rapidly became popular because it was cheaper to instal and operate than the old jig processes, and required less attention, labour and floor space. To-day it has a serious rival in the Rheolaveur process. The Rheolaveur washer has the advantage that it can clean coal below $\frac{1}{16}$ in., a size which is often the dirtiest, and which cannot be treated at all by the Chance method. The Deister-Overstrom table is also becoming popular for anthracite cleaning, but the Chance and Rheolaveur process have the advantage that they can handle sizes up to 4 in., and require less floor space, power and attention.

In the Wyoming district the pea and buckwheat sizes (through $\frac{11}{16}$ in.) were fairly clean, and only recently has washing been instituted. Usually the larger sizes (over $\frac{11}{16}$ in.) are cleaned on spirals, and the smaller sizes (through $\frac{11}{16}$ in.) are either uncleaned or are washed in jigs. In the Lehigh and Schuylkill districts, hand-picking is usual for sizes above $3\frac{7}{16}$ in., and jig washers have been used for 30 to 40 years for the sizes $3\frac{7}{16}$ in. to $\frac{1}{4}$ in., and occasionally for sizes 6 to $3\frac{7}{16}$ in.

Complete figures for the amount of anthracite cleaned or washed cannot be given, but the proportion of the total production is probably high. For example, there were 250 Elmore jigs, each with a capacity of perhaps 40 tons per hour, installed in Pennysylvania from 1903 to 1921. On the basis previously assumed, the estimated annual capacity of these washers is at least 20,000,000 tons, or one

quarter of the total anthracite production. In 1923, 4 million tons of anthracite were recovered from culm-banks and washed.

In America the preparation of anthracite requires more care than the preparation of bituminous coals, for, apart from the fact that in certain districts the anthracite seams are dirtier than the bituminous coal seams, nearly two-thirds of the output is used for domestic purposes, for which higher standards of purity are required. Table 58 shows the amounts of impurity which are allowed in different sizes of anthracite for sale to specification (Coal Catalog. Pittsburgh, Pa., 1925).

TABLE 58.—PERMISSIBLE AMOUNTS OF DIRT AND BONE COAL IN DIFFERENT SIZES OF AMERICAN ANTHRACITE

Name.	Size (in.).*	Per cent.	
		Dirt.	"Bone" Coal.
Broken (or grate)	4 –2¾	1	2
Egg	2¾–2	2	2
Stove	2 –1⅜	4	3
Chestnut	1⅜– ¾	5–7	5
Pea	¾– ½	8	10
Buckwheat	½– ¼	10	10

The figures in Table 59 give the numbers of washing plants in the State of Washington, U.S.A., up to the end of 1924 (Macmillan and Bird, Bull. 28, Univ. of Wash. Eng. Exp. Stat.). In 1923, 907,339 tons of coal were washed in this State, amounting to 24·7 per cent. of the total output. The other figures give the numbers and types of washers (including pneumatic tables) erected in the anthracite and bituminous coalfields of U.S.A. during 1926 (Frazer, *Coal Age*, 1927, 31, 33).

It will therefore be seen that, since its introduction in 1920, the Deister-Overstrom table has made rapid progress, almost to the exclusion of other types of concentrating tables. These figures show that concentrating tables are used considerably in U.S.A. In the State of Washington nearly half the plants include concentrating tables.

From 1903 to 1921 about 100 Elmore jigs were erected in the bituminous coalfields. Assuming a capacity of 40 tons per hour for each jig, the estimated annual washing capacity of these jigs would be 8 million tons, or about one-third of the actual amount of bituminous coal washed.

* Square holes.

Type of Washer.	No. of Washers in Washington, U.S.A.	No. of Washers Erected in U.S.A. during 1926.	
		Anthracite Mines.	Bituminous Mines.
Jigs, miscellaneous	48	8	2
Cone washers	4	—	2
Concentrating tables :			
Deister-Overstrom	12	11	2
Deister Plat-O	4	—	—
Massco	2	—	—
Overstrom Universal	3	—	—
Chance	—	12	1
Rheolaveur	—	6	2
Hydrotator	—	2	—
Pneumatic tables	—	—	13

Present Position in the British Empire

Large quantities of coal are produced in a number of the British Dominions, the coal production (excluding lignite) in 1925 being: India, 20·2 million tons ; Australia, 14·9 million tons ; South Africa (including Rhodesia), 14·5 million tons ; and Canada, 8·6 million tons. The numbers of plants erected by European coal-washery constructors is very small, only the Baum washer having been built in any numbers (four in South Africa and three in Canada). One Meguin washer has been built in Australia. India has apparently none of the washers popular in Europe, but it is known that Indian coals do not lend themselves to ordinary coal-washing methods, because they have a very high percentage of fixed ash.

CHAPTER VII

THE HUMBOLDT JIG WASHER

THE Humboldt Engineering Company was formed in 1870, incorporating a smaller firm, Sievers & Co., which had been building mining machinery in the Ruhr district of Germany since 1856. A Sievers' jig built in 1860 is illustrated in Fig. 26. It consisted of a short but broad-limbed U-shaped vessel, in one of the limbs of which the plunger worked, and the sieve was placed horizontally in the other. The plunger, of rectangular cross-section, received its impulses from a rotating shaft by means of a disc-crank lever system, which made the downstroke quicker than the upstroke. Water was continuously admitted to the wash-box through the main, *h*, the water flow being regulated by means of the valve, *i*, which was controlled by a hand-lever. During the upstroke of the plunger, the inflow of water helped to reduce the effect of suction on the bed above the sieve, and, during the downstroke, the water flow carried the upper layers of material from the bed through the openings, *a*, set at the four corners of the washing compartment, into the channels, *b*, from which the material was carried to the collecting trough, *c*. The dirt was removed through two openings, *u*, in the sieve, into the dirt pipes, *r*, which discharged into the channel, *l*. The dirt removal was governed by means of hand-levers, t, t_1. The finest dirt which passed through the sieve was removed through the bottom opening, *g*, at intervals, by the manipulation of another lever.

It may be noted that the general design of the wash-box was superior to that of many of the other early jig-washers in that the plunger and sieve areas were well proportioned, the water currents produced by the plunger were not throttled, and the shape of the bottom portion of the jig aided the attainment of uniform water currents over the whole area of the sieve. The disadvantage of this form of jig was the slow rate at which the dirt could be removed, on account of the small dimensions of the exit ports, and the washing capacity was therefore limited. The area of the sieve used in the wash-box illustrated in Fig. 26 was only 0·84 metres by 0·68 metres (2 ft. 9 in. by 2 ft. 2 in.), and the number of strokes made was 41 per minute. Attempts were made to increase the capacity of the washer by increasing its size and by the construction of a double washer (1868), in which a sieve was placed on either side of the plunger compartment.

The Humboldt Company—after the incorporation of the Sievers

firm in 1870—still built wash-boxes of rounded cross-section. Towards the end of the nineteenth century, the design was altered and the form of wash-box illustrated in Fig. 27 was adopted. In this

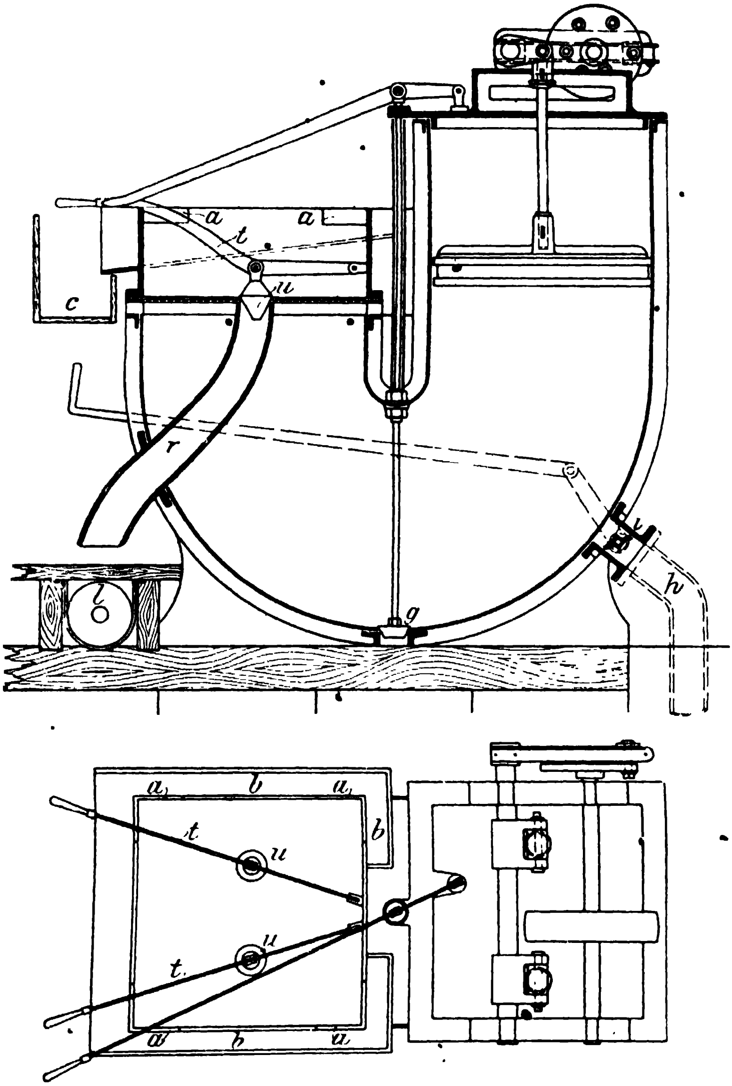


FIG. 26.—Cross-Section and Plan of Early Sieves Wash-box.

design, the lower portion of the wash-box was made of rectangular pyramidal shape, the weight of the wash-box being taken by two supporting beams at the sides. The ratio of the plunger to the sieve area was reduced from 1 to 1 as used in the Sievers' jig, to 3 to 4 or

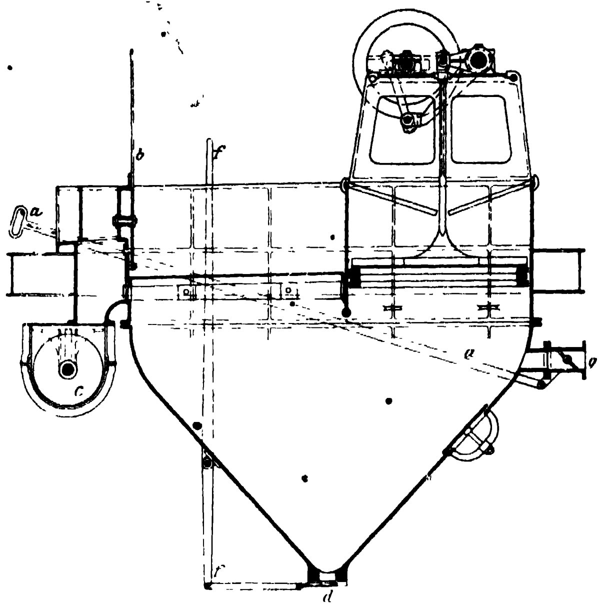


FIG. 27.—Cross Section of Early Form of Humboldt Nut-Coal Wash-box.

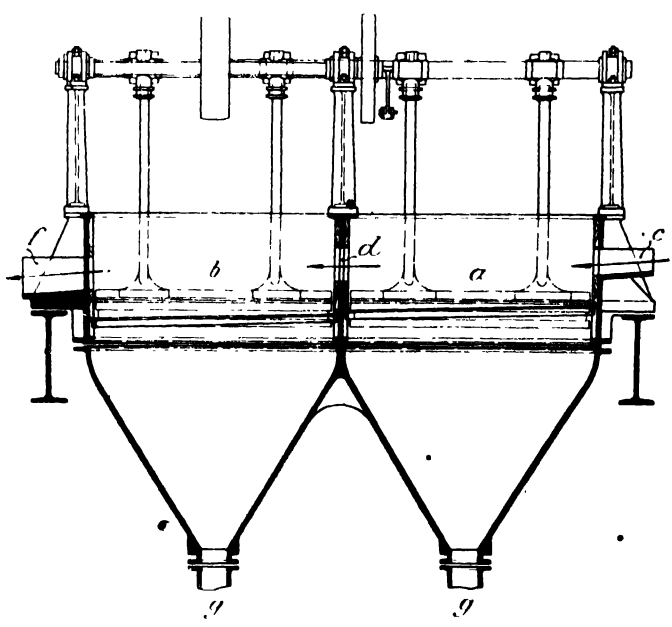


FIG. 28.—Longitudinal Section of Early Form of Humboldt Fine-Coal Wash-box.

5 to 6 for nut-coal washers, and 4 to 5 for fine-coal washers. The sieve was inclined downwards towards the front of the wash-box, and a number of horizontal slots were provided in the front plate

for the removal of the dirt. The uncovering of the slots was governed by means of a hand-lever, *b*. The water flow was similar to that in the earlier Sievers' jig, the valve, *g*, being governed by a hand-lever, *a*. The dirt was carried by a stream of water through the slots in the front plate of the box into the channel, *c*. The finest dirt passing through the sieve was removed through the opening, *d*, governed by means of the hand-lever, *f*. The raw coal was admitted at the back of the wash-box near the partition between the two compartments of the box.

For the fine-coal washer, the general design of the wash-box was similar to that of the nut-coal wash-box, but two units were joined together, and the direction of flow of the coal through the wash-box was parallel to the division plate between the sieve and the plunger compartments, the method which Lührig introduced in his fine-coal washer (see Fig. 28). Thus the speed of travel of coal through the wash-box was slower than in a nut-coal wash-box, and the coal was subjected to a greater number of pulsations during its passage through the box.

A layer of feldspar was placed on the sieve and the apertures of the sieve were large enough to allow the dirt to pass through. Thus, as in all feldspar washers, no dirt removal valves were necessary, and the dirt settled to the bottom of the pyramidal portion of the box which was connected to a bucket elevator.

For both types of washer a "knee-lever" mechanism was employed to give a slower movement of the plunger in the upstroke than in the downstroke. This mechanism is illustrated in Fig. 29. It consisted of a revolving shaft, *a*, to which was fixed a crank, *f*, which was connected by means of a link, *c*, to a bent lever, *d*. At its other extremity the bent lever was connected to a rocking shaft, *b*, which carried a crank, *e*, to which the eyes of the plunger rod were fastened by a pin. In Fig. 29 a side view of the revolving shaft crank and the connecting link to the bent lever is shown. The crank pin connecting the revolving shaft crank and the link could be adjusted within the limits of the slot, *h*, to vary the diameter of the circle travelled by this crank pin, and so vary the length of the stroke made by the plunger. In the position of the mechanism illustrated in Fig. 29 the rocking-shaft crank pin, and consequently the plunger, is in its highest position. As the revolving-shaft crank, *f*, moves in a clockwise direction the rocking-shaft crank pin is depressed. When the revolving-shaft crank pin is in the position, B, the rocking-shaft crank pin is in its lowest position, B_1. As the revolving-shaft crank pin continues to descend, the rocking-shaft crank pin rises, and reaches its highest position, T_1, when the revolving-shaft crank pin attains its highest position, T. As the revolving shaft is driven by a pulley at a constant speed of rotation, the plunger is depressed during a third of the time taken by the revolving-shaft crank pin to complete a revolution. The time taken for the downstroke of the plunger is therefore one-half of the time taken for the upstroke,

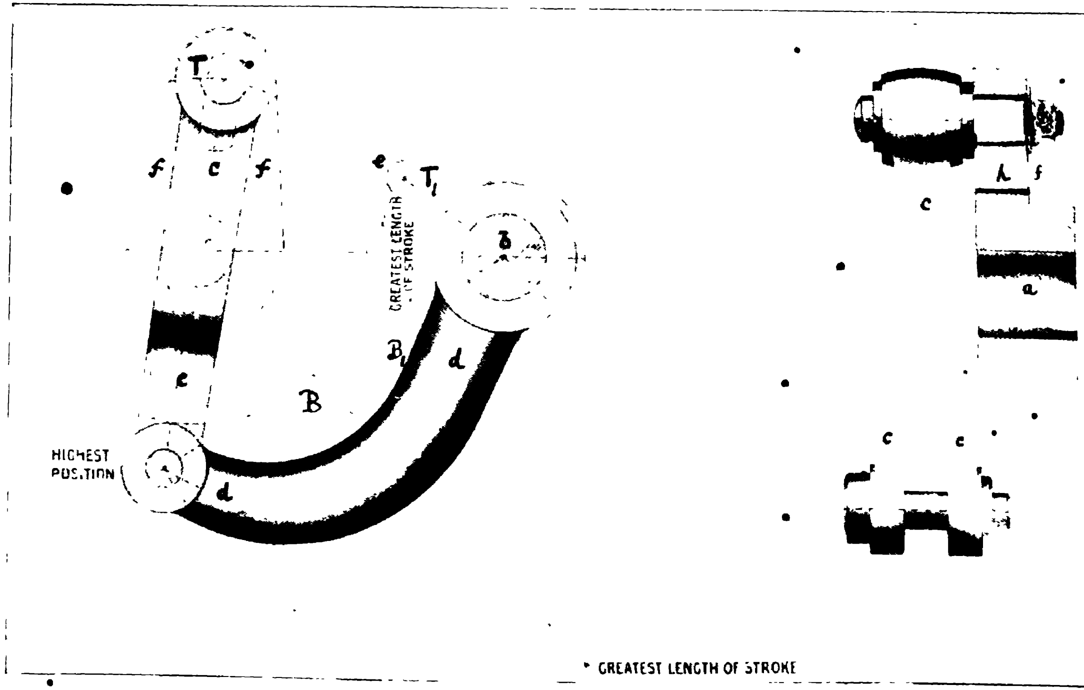


FIG. 29.—Knee-lever Differential Mechanism.

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and the effect of the mechanism is to make the velocity of the upward water currents through the sieve twice the velocity attained by the downward water currents.

The ratio of the velocities of the downward and upward water currents through the bed of the jig was further reduced by admitting water constantly to the jig. The effect of the continuous addition of water was to increase the velocity of the upward current during the downstroke of the plunger. During the upstroke of the plunger, when there was no overflow from the box, the added water tended to compensate for the upward displacement of the plunger, so that the amount and velocity of the water passing down through the sieve was reduced.

The use of similar differential mechanism was very popular in the earlier coal-washing jigs, though less use was made of them in later types. In ore-dressing practice, the " knee-lever " mechanism was first applied to the washing of small-sized materials, and was later applied to the washing of coarse-grained materials. In coal-washing practice, on the other hand, the " knee-lever " mechanism was first applied to nut-coal jigs and was later used for fine-coal jigs. The " knee-lever " mechanism was, however, used, instead of the eccentrics, in the Humboldt washer for fine coal. With the use of an eccentric in the Sievers jig, the times taken for the upward and downward strokes of the plunger were the same. Consequently, the difference between the upward and downward water currents through the bed on the sieve, which is obtained when using a differential mechanism, was absent. In the latter case any differentiation obtained was due to the water flow alone.

It was the custom in Humboldt washeries built in the latter part of the nineteenth century to divide the raw coal into a number of fractions before washing. These fractions, which in one instance quoted, for a washery erected at Blanzy, France, in 1898, of 80 tons per hour capacity, were six in number, namely, 60 to 45 mm. ($2\frac{3}{8}$ to $1\frac{3}{4}$ in.), 45 to 25 mm ($1\frac{3}{4}$ to 1 in.), and 25 to 10 mm. (1 to $\frac{3}{8}$ in.) for the nut-sized coal, and 10 to 7 mm. ($\frac{3}{8}$ to $\frac{1}{4}$ in.), 7 to 3 mm. ($\frac{1}{4}$ to $\frac{1}{8}$ in.), and 3 to 0 mm. ($\frac{1}{8}$ to 0 in.) for the fine coal. These fractions were washed in five nut-coal wash-boxes and five fine-coal wash-boxes, whilst the crushed large dirt was rewashed.

After Baum had shown that it was not necessary to divide the coal into so many fractions before washing, the number of wash-boxes used in a Humboldt washery was reduced, though the practice of sizing before washing was still employed. It was also found possible to wash fine coal successfully without the use of a feldspar bed.

In a Humboldt washery of more modern construction, washing 125 tons per hour, the unwashed coal arrives at the washery in wagons and is unloaded into the elevator pit, A, Fig. 30. From this point the raw coal is elevated to the top of the building and is delivered to the screens for classification. In this example of a

Humboldt washery the coal is brought to the top of the washery by two elevators and is discharged on to one of the fixed sieves, B. The raw coal is split into two fractions, on and through a 7 mm. (0·28 in.) screen. The coal less than 7 mm. size is collected by enclosed launders under the fixed sieves, and is passed to the fine coal bunker, H, Fig. 31. This fine coal is then carried by an elevator, V, from the bottom of the fine-coal bunker and is discharged on to a screw conveyor, I, which distributes the coal over the dust screens, a plan of which is given in Fig. 32. From this

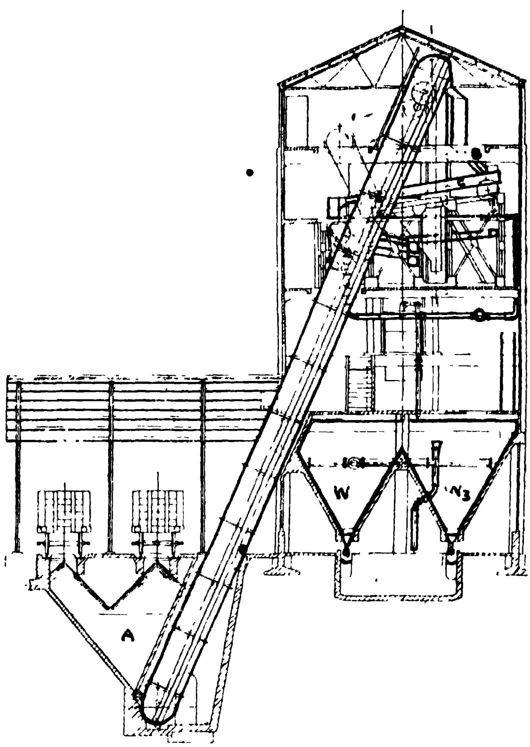

FIG. 30.—Cross Section through Humboldt Washery.

illustration it will be seen that there are twenty-two screens arranged in two rows of eleven on either side of the central screw conveyor, I, Figs. 31 and 32. The screens are gently inclined away from the screw-conveyor and are vibrated by ratchet wheels working on a shaft which passes under each row of eleven screens and near to the screw conveyor. The fine coal works its way down the sieves, which are of 1 mm. ($\frac{1}{25}$ in.) mesh, so that most of the fine dust less than this size passes through the sieves and is carried by the screw-conveyor, J, Fig. 31, to the dust hopper, K, from which it may be mixed with the washed coking slack, if of suitable quality. The

fine coal passing over the dust screens drops into a trough inclined downwards towards the middle of its length, and is carried by a stream of water, entering at each end of the trough, into a launder,

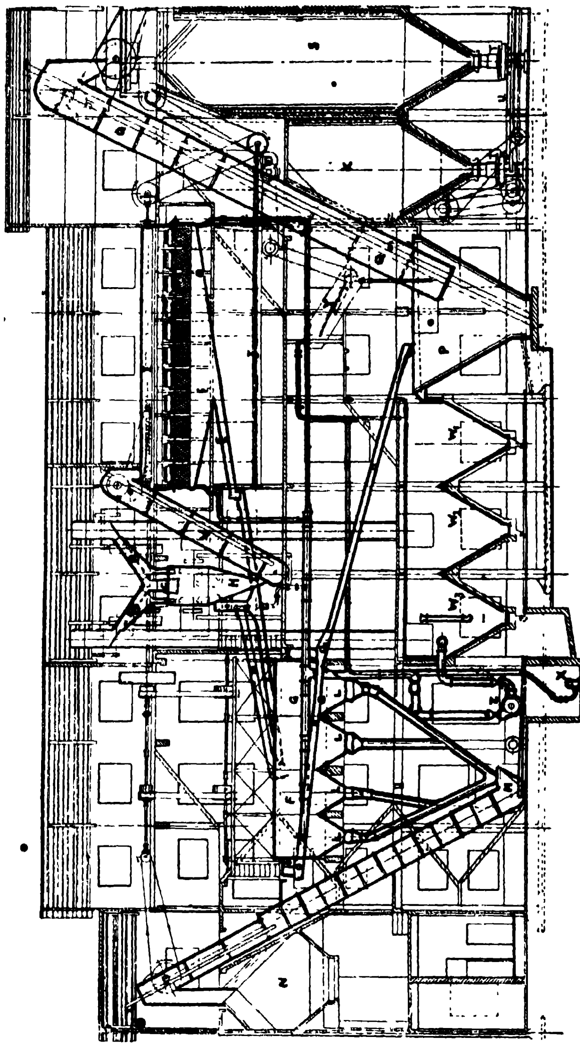

FIG. 31.—Longitudinal Section through Humboldt Washery.

E, Figs. 31 and 32, and thence to the fine coal wash-box, G, Fig. 31.

The coal which has passed over the 7 mm. screen is discharged on to the 11 mm. (0·43 in.) shaking screen, C, Fig. 30, and the large coal passing over this screen is carried by a stream of water in the launder, D, Fig. 31, to the nut-coal wash-box, F. The coal passing

Humboldt washery the coal is brought to the top of the washery by two elevators and is discharged on to one of the fixed sieves, B. The raw coal is split into two fractions, on and through a 7 mm. (0·28 in.) screen. The coal less than 7 mm. size is collected by enclosed launders under the fixed sieves, and is passed to the fine coal bunker, H, Fig. 31. This fine coal is then carried by an elevator, V, from the bottom of the fine-coal bunker and is discharged on to a screw conveyor, I, which distributes the coal over the dust screens, a plan of which is given in Fig. 32. From this

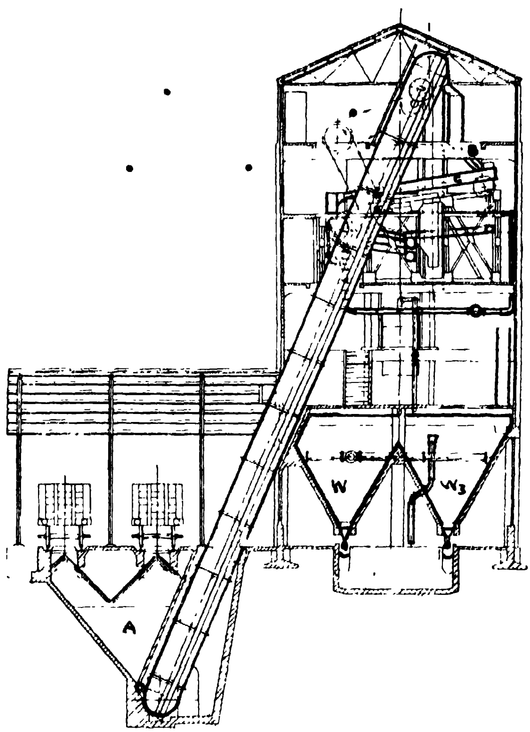

Fig. 30.—Cross Section through Humboldt Washery.

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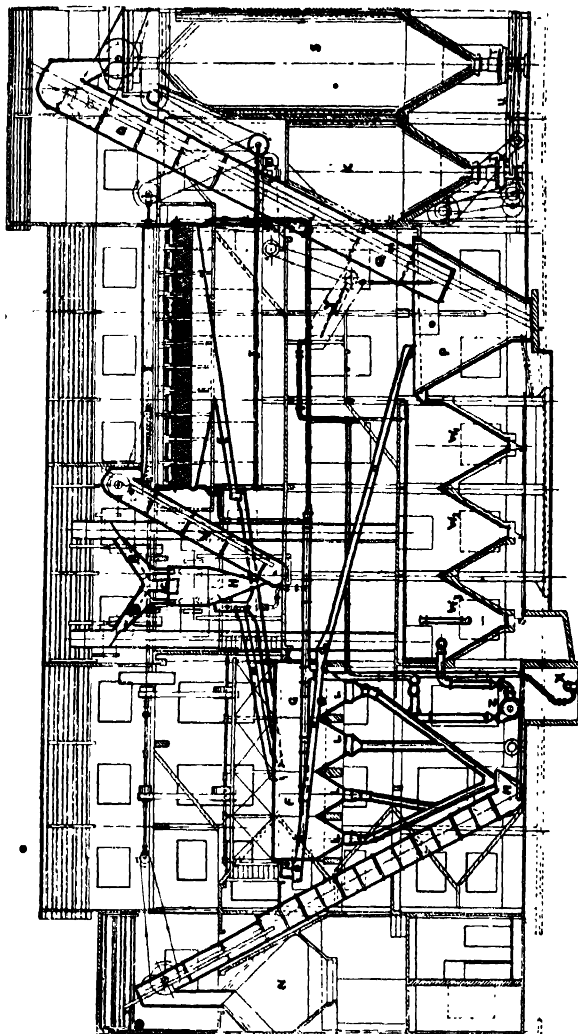

FIG. 31.—Longitudinal Section through Humboldt Washery.

E, Figs. 31 and 32, and thence to the fine coal wash-box, G, Fig. 31.

The coal which has passed over the 7 mm. screen is discharged on to the 11 mm. (0·43 in.) shaking screen, C, Fig. 30, and the large coal passing over this screen is carried by a stream of water in the launder, D, Fig. 31, to the nut-coal wash-box, F. The coal passing

through the 11 mm. screen (being already greater than 7 mm. in size) is carried into the launder, E, where it joins the coal smaller than 7 mm. coming from the dust-screening plant. The mixed

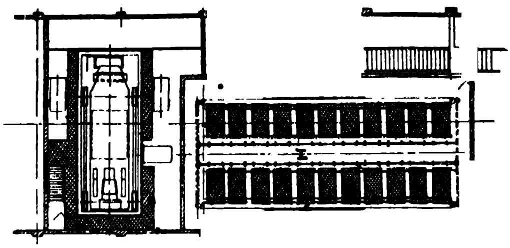

FIG. 32.—Plan of Dust Screens.

sizes pass to the fine-coal wash-box, G. The sizing of the coal before washing may be summarised as follows :—

Raw Coal.
7 mm. sieve
through — on
1 mm. sieve — 11 mm. sieve
through — on — thro' — on
0-1 mm. 1-7 mm. 7-11 mm. over 11 mm.
Dust hopper — Fine-coal washer — Nut washer

In this way the size ratio of the coal passing to the nut-coal wash-box is $\frac{2{\cdot}5}{0{\cdot}43} = \frac{6}{1}$ (assuming that the largest size of coal treated is $2\frac{1}{2}$ in.), and the ratio for the coal passing to the fine-coal wash-box is $\frac{0{\cdot}43}{0{\cdot}04} = \frac{11}{1}$.

In the washer illustrated in Figs. 30 and 31, the nut and the fine-coal wash-boxes are placed end to end for convenience in control. From Fig. 31 it will be seen that the nut coal is carried along by the stream of water in the launder, D, to the wash-box, F, whilst the fine coal enters the wash-box, G, through the launder, E. The points of admission of the coal are thus at adjacent ends of the two wash-boxes, and the flow of the horizontal water currents in the two wash-boxes is in opposite directions. The washed nuts are discharged into the launder, O, and are later joined by the washed

FIG. 33.—View of Top of Drainage Elevators, Humboldt Washery.

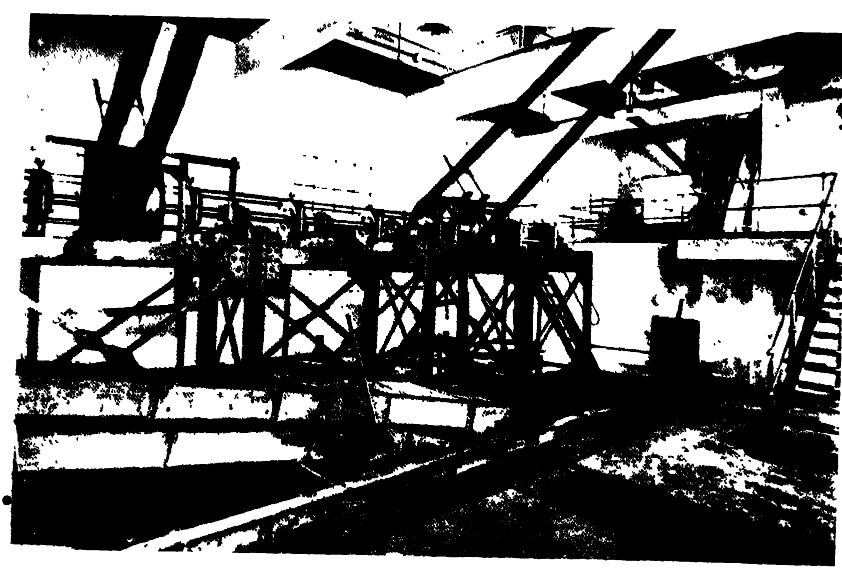

FIG. 34.—View of Humboldt Wash-boxes.

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fine coal at O_1, the combined products being carried into the dredging sump, P. From this sump the coal is elevated by means of the drainage elevator, Q, which consists of an endless series of perforated trough-shaped buckets connected together by means of link chains working on inclined guides. A view of a drainage elevator is shown in Fig. 33. The buckets move upwards at a slow speed which permits a certain amount of drainage before the contents of the buckets are delivered into a shoot, from which they pass to a conveyor and to the drainage bunkers, S, Fig. 30. After drainage in these bunkers for a suitable period, the washed slack is conveyed to a storage bunker in connection with the coking plant, or it may be loaded into wagons. The circular table, which forms the base of each drainage bunker, is rotated, and the coal is guided by a "plough" on to a moving rubber belt, which conveys the coal to the storage bunker or to the wagon-loading shoots.

The water which has carried the washed coal into the dredging sump, P, overflows from here into a series of spitzkasten, or slurry settling tanks, W, W_1, W_2, W_3, etc., Figs. 30 and 31. The fine particles of coal and dirt which are unable to settle in the dredging sump, P, because of disturbances caused by the influx of the washed coal and the movement of the drainage elevator, settle in these tanks, the coarsest particles of the slurry in the first tank, W_1, and the finer particles in the successive tanks, W_2, W_3. The clearest water is pumped by the pump, Z, and circulated through the mains to the various launders, D, E. An overflow pipe is fitted in the settling tank, W_3, Figs. 30 and 31, in case of failure of the main pump.

The slurry settled in the spitzkasten is periodically run off through the bottom openings into channels (see Fig. 30), and to a slurry sump, whence it is pumped by the centrifugal pump, X, Fig. 31, on to a fine mesh sieve, Y. This sieve filters out all save the finest particles, which, with the bulk of the water, passes through the sieve into the dredging sump, P. The coarsest particles of slurry are washed on to the top of the buckets of the drainage elevator, Q.

Some of the refuse passes through the sieves of the wash-boxes, but most of the dirt collects on the sieve and is periodically allowed to run through into the refuse-collecting launders, by the release of the gate covering the refuse ports. The rejected refuse settles in the pyramidal bottoms of the wash-boxes, L, and thence passes down the refuse-collecting pipes which unite at the boot of a refuse elevator, M. The refuse is elevated and discharged into a refuse hopper, N. The elevator is totally enclosed and filled with water up to the level of the water in the wash-boxes, so that the refuse may be continually removed in a closed circuit.

The nut-coal wash-box and the fine-coal wash-box both consist of two compartments, in each of which the coal is subjected to a jigging motion. A side view of one compartment is shown in Fig. 35,

in which s is the fixed sieve, and p the plunger, operated by means of the adjustable eccentric, e. The washed coal flows over a weir, and the dirt is removed by raising the gate covering the refuse ports. The dirt passes through the ports to launders leading to the collecting cone below the sieve. In each of the two compartments of one wash-box there are two plungers working independently, so that the coal is subjected to a number of pulsations in its passage through the box, and ample opportunity for stratification is provided.

In the nut-coal wash-box, the fixed sieve in the first compartment is a 12 mm. ($\frac{1}{2}$ in.) screen, so that the apertures are slightly larger than the size of the smallest particles washed in that wash-box. In the second compartment of the nut-coal wash-box, the fixed sieve is an 8 mm. ($\frac{5}{16}$ in.) screen. The two compartments of the nut-coal wash-box are of the same size, namely, 5 ft. 5 in. by 5 ft. 10$\frac{1}{4}$ in., so that the total length of travel of the nut coal through the wash-box is 10 ft. 10 in. The number of strokes per minute made by the plungers in each compartment of the nut-coal wash-box is fifty-six, but in the first compartment the length of the stroke is 7 in., compared with 5$\frac{1}{2}$ in. in the second compartment.

The fine-coal wash-box is of the same width as the nut-coal wash-box, but the length of each compartment is increased to 7 ft. 1 in., and the total length of travel of the fine coal is 14 ft. 2 in. The size of aperture in the fixed sieve of this wash-box is the same in each compartment, namely, 5 mm. ($\frac{3}{16}$ in.), so that the finest particles of dirt may pass through the sieves. The bed is therefore left in a more open condition, approximating to that found in a fine-coal wash-box using a false bed of feldspar, but consisting actually of the coarser particles of dirt. In each compartment of the wash-box 40 strokes are made per minute, the length of stroke being 4 in. in the first compartment and 3 in. in the second compartment.

The average results of working over a period, in the washer described, showed that 2·5 per cent. of free dirt remained in the washed coal, and 2·75 per cent. of free coal passed away with the dirt.

When the number of strokes per minute, and the length of each stroke in each compartment, as well as the quantity of water in circulation, have been fixed, the conditions which remain under the control of the washeryman are the rate of feed of the coal, and, more particularly, the rate at which the refuse is removed. In this washery, in the absence of storage bunkers to enable a uniform rate of feeding of the raw coal to be made, an adjustment is made of the amount of coal entering the bottom of the raw coal elevator from the elevator pit. This is made possible by the use of a slide, over the opening into the elevators, worked by a spindle and hand-wheel, the latter being situated on the wash-box floor so that it may be adjusted by the washeryman. Although this method of using a slide in the elevator pit does not allow the same degree of control as the use of a bunker set at the top of the raw-coal elevators (on account of the interval of time elapsing between the discharge

of the contents of the elevator after an adjustment of the bottom slide has been made), it has the advantage that it obviates the cost of bunkers, and so is frequently used.

Assuming a regular rate of feed of raw coal to the wash-boxes, a layer of refuse accumulates on the sieves and constitutes what is called the "bed" of the washer. The thickness of the bed is controlled by removing a quantity of refuse periodically, by lifting the lever, *l*, Fig. 35, and uncovering the refuse-removal ports. In two-compartment jigs (such as those described in this chapter), there are, in each wash-box, two refuse ports with suitable gates and levers. A view of the two wash-boxes, shown diagrammatically in Fig. 31, is given in Fig. 34. In the first compartment of the jig the heavier dirt settles to the bed, and the lighter dirt is mostly removed in the second compartment. It is therefore frequently the practice to leave the refuse gates of the first compartment slightly raised to allow a constant removal of part of the heavier dirt. This is arranged by setting a pin through a hole in the lever, *l*, Fig. 35, and through a quadrant in front of which the lever moves, thus fixing the extent of the opening of the refuse gate as desired. If the rate of feed is constant, and the percentage of dirt in

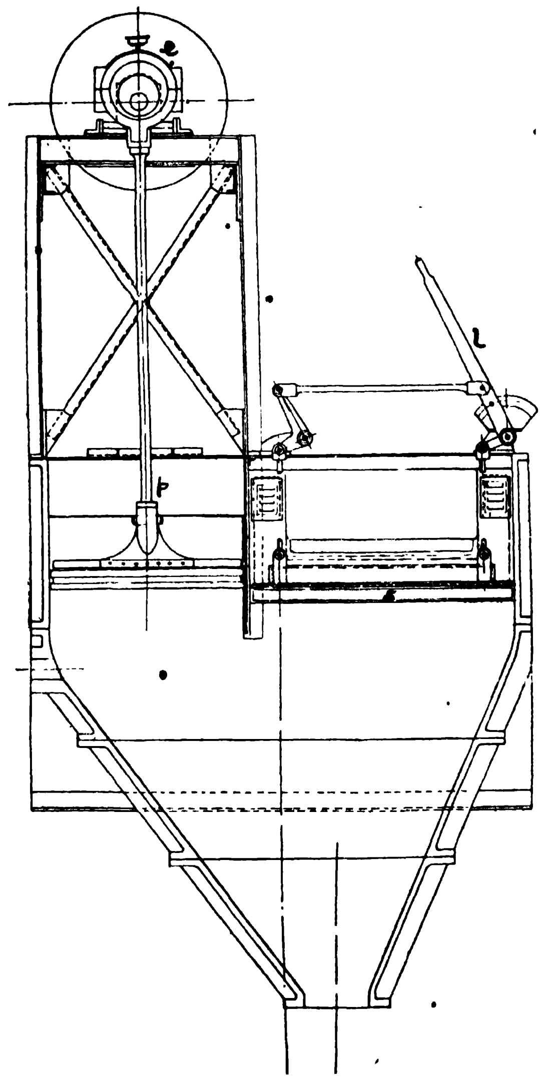


FIG. 35.—Cross Section of Humboldt Wash-Box.

the raw coal does not vary greatly, it is possible by this means to maintain an almost constant depth of dirt bed in the first compartment. If the depth of the bed increases too rapidly, the excess may be removed by opening the refuse port fully at intervals.

In the second compartment of the jig, where the lighter dirt settles, the thickness of the bed is more usually governed by intermittently opening the refuse ports. The conditions in this bed will therefore vary more than in the bed of the first compartment, and it is in the control of these variable conditions that the skill and conscientiousness of the washeryman are important. To determine the thickness of the bed the washeryman may use a light rake or other convenient tool, and rest it on the top of the material in the wash-box, and, whilst exerting a gentle pressure, allow the rake to move upwards and downwards with the movement of the water currents. In a short time the rake will pass through the layers of coal and will rest on the bed, which will offer a greater resistance to movement. By exerting a greater pressure it is possible to pass through the bed and reach the fixed sieve on which the bed rests. Thus it is possible to measure the thickness of the bed; when it is too thick it is said to be "heavy," and it is "light" when the bed is not sufficiently thick.

In the Humboldt washer there are also small pressure-release compartments fitted with slots at the front and back of the wash-box before each refuse gate (see Figs. 34 and 35). These compartments are in communication with the fixed sieve. When the bed is not "heavy" the impulse of each downward stroke of the plunger is fairly uniformly expended throughout the area of the bed. When the bed becomes "heavy" its resistance increases, and the water currents may not be sufficiently strong to pass through the bed. Release of the pressure is then obtained through the pressure-release compartments. This pressure-release causes water to be spouted from the slots of the compartment and draws the washeryman's attention to the state of the bed.

If the bed is too thin, light dirt particles may be raised by the upward water currents and be carried over with the washed coal. If the bed is too thick the upward currents of water are insufficient to overcome the resistance of the bed, and coal particles may sink and be carried away with the refuse, and removal of dirt with the washed coal may also occur. If the rate of feeding of the raw coal is too great, the coal, on entering the wash-box, may pack solidly on the bed and rise above the surface of the water, making the compactness of the freshly-fed coal excessive and preventing that free movement of the coal and dirt particles which is necessary for separation. This state of affairs is usually only found to occur near the end of the wash-box at which the feed enters, because, as the coal travels towards the discharge end, the upward currents break up the compact mass and permit stratification of coal and dirt particles. If the first upward current of water be employed

in breaking up the compact mass, it is prevented from effecting any separation of coal and dirt; consequently, there may be one less pulsation in which to bring about stratification, and the efficiency of washing may be reduced. Furthermore, if the refuse gates have been recklessly opened to meet the sudden increase of feed, coal may be found in the dirt, or, as is more common, dirt may be carried over with the washed coal.

One of the most important factors in maintaining efficiency of washing, apart from the human factor involved in the control of the refuse gates, is the maintenance of a constant feed of raw coal to the wash-boxes. The regularity of feeding may be interfered with by external conditions, such, for example, as the irregular arrival of wagons of raw coal to the washery, or by a shortage of wagons to remove washed nuts or refuse, for which only small bunker space may be provided. Internal conditions such as failure of power, mechanical failure of the main washery pump, or of other essential portions of the plant, may also interfere with regular working. On restarting after temporary stoppages it is necessary to recover a suitable working bed, and, during this period, dirt particles may be carried over with the washed coal. A further disadvantage of a stoppage is the settling of slurry in the dredger sump so that, on restarting the plant, large quantities of settled slurry are brought up by the elevators *en masse*, instead of being distributed evenly and admixed with coarser coal. It is well known that an aggregation of slurry, containing a larger quantity of ash and water than an average sample of washed coal, when included in a charge to a coke oven, for example, interferes seriously with the regular discharging of ovens, and gives an inferior coke product containing much breeze. Thus it is most important for the regular working of a washery to have well-designed sidings and to have an efficient system of traffic control, whereby shortage of wagons of unwashed coal, or of empty wagons to remove the products, may occur only rarely.

Among other circumstances which lead to inefficient washing are those which give abnormal conditions in the wash-boxes. Thus tramp iron, consisting of nails, bolts, washers, pick-heads and the like, accumulate to a surprising extent in the nut-coal wash-box, having, of course, made its entry with the raw coal. Such material may lodge under the refuse gate whilst it is open and prevent its complete closure. Should this not be noticed by the washeryman, the bed will soon be lost and the washing will be imperfect. Lodgment of pyrites under the refuse gate may also cause trouble. The tramp iron may cause further trouble by damaging the fixed sieve and allowing escape of the bed. If a sieve is badly damaged the washer must be shut down until the sieve is repaired. Further trouble may be experienced by the perforation of the screens used for the preliminary classification of the raw coal, in which event larger sized coal particles may pass to the fine-coal washer, where

the current velocities may be insufficient to float them, so that they would pass away with the refuse.

The introduction of dust-screens to remove all coal below about $\frac{1}{25}$ in. is a good feature of Humboldt practice. The finest sizes of coal, below about $\frac{1}{50}$ in., do not obey the normal laws of fall in water, and separation of coal from dirt in such sizes is almost impossible by means of jigs. If a fine-coal washer is adjusted to float coal up to $\frac{3}{8}$ in. size, small dirt particles may be kept in suspension, and fine coal particles may be carried down into the bed during the suction stroke of the plunger.

The size of coal which may be conveniently screened out will vary according to the type of coal washed. The larger the size of screen used the easier is the screening operation, but the greater is the proportion of coal not washed. It is desirable, therefore, to remove only the smallest dust. When, however, the coal fed to the washery arrives in a moist condition, dry screening of small sizes becomes impracticable owing to the blocking up of the screen openings, and the dust screens must be by-passed. If it is possible to use them and to remove the finest dust, the efficiency of drainage of the washed coal is greatly increased. Where a washery is erected in conjunction with a coke oven plant, the lower moisture content of the coal used greatly increases the throughput of the ovens, besides decreasing the slurry evils.

Some results of washing in a Humboldt washer have already been given. Further results were recorded by J. W. Lee (*Gas World*, 1917 (Coking Sect., Dec.) 14) for a washer which was fifteen years old. The raw coal was screened into sizes, $\frac{3}{4}$ to $\frac{1}{2}$ in., $\frac{1}{2}$ to $\frac{1}{4}$ in., $\frac{1}{4}$ to 1 mm., before washing, the dust less than 1 mm. being removed from the smallest size of coal by shaking screens. The dirt removed from the two smaller sizes, $\frac{1}{2}$ to $\frac{1}{4}$ in. and $\frac{1}{4}$ in. to 1 mm., was rewashed. The results are given in Table 60.

TABLE 60.—RESULTS OF WASHING IN A HUMBOLDT WASHER

Size.	Unwashed Coal.			Washed Coal.			Refuse, Coal per cent.	Re-washed Refuse, Coal per cent.
	Dirt per cent.	Ash per cent.	Sulphur per cent.	Dirt per cent.	Ash per cent.	Sulphur per cent.		
$\frac{3}{4}$–$\frac{1}{2}$ in. . .	19·3	11·6	2·3	3·2	3·7	2·1	3·3	—
$\frac{1}{2}$–$\frac{1}{4}$,, . .	24·7	19·5	2·4	3·8	5·2	2·2	22·2	3·9
$\frac{1}{4}$ in.–1 mm. . .	32·8	26·6	3·0	8·4	8·9	2·4	8·8	3·1
Dust <1 ,, .	33·3	22·4	2·6	Not washed.			—	—

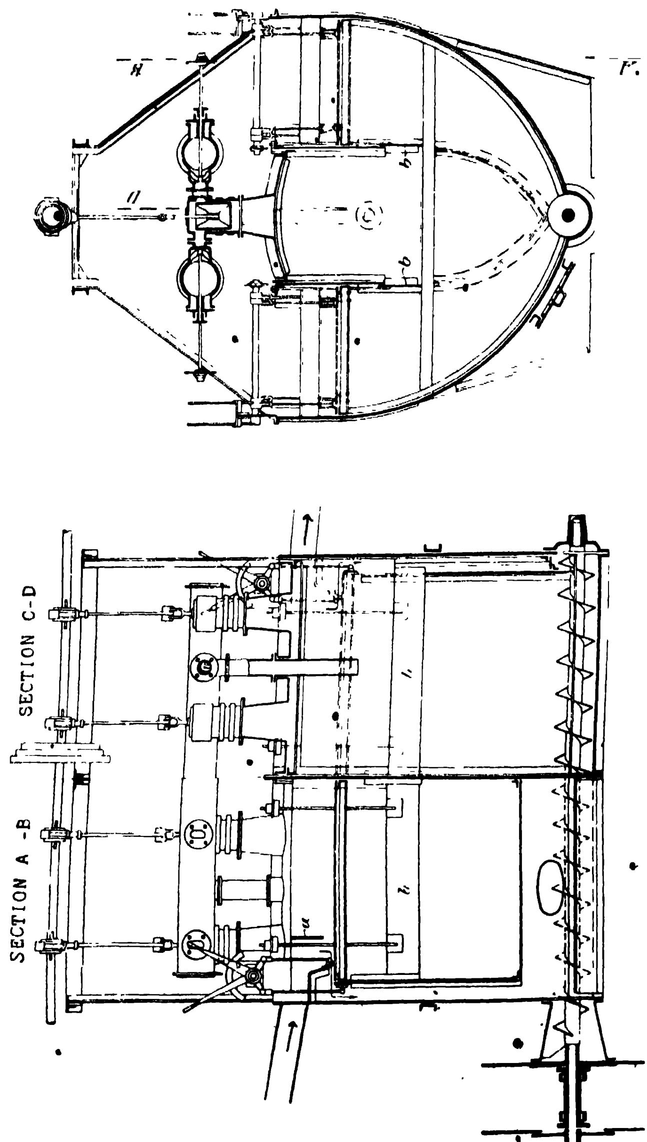
CHAPTER VIII

THE BAUM JIG WASHER

In 1892 Baum introduced his well-known jig washer, in which the water pulsations were obtained by admitting compressed air above the water in one section of the box. Apart from this novel means of obtaining the jigging motion, the methods of dealing with the coal in this first Baum washer were similar to those employed in other jig coal-washers of the period, that is, the raw coal was sized before washing into a number of fractions, each of which was treated in a separate wash-box. For each box the length and number of the strokes made, the size of sieve mesh, and the quantity of water admitted, were separately regulated. For example, at Zeche Carl Colliery, Kölner Bergwerks-Verein, Altenessen, in 1898, coal up to 80 mm. ($3\frac{1}{8}$ in.) was divided into four nut fractions and one fine coal fraction ; each of the nut sizes of coal was treated in a separate wash-box, and five boxes were required for the fine coal, making nine boxes in all. At Zeche Horstfeld Colliery, the length of stroke of the valve piston was 70 mm. ($2\frac{3}{4}$ in.), and the number of strokes made per minute was 42 to 60 for the two larger-sized fractions of nuts, and 75 to 109 for the two smaller-sized nut fractions ; the sheet steel sieve had perforations of 6 to 16 mm. ($\frac{1}{4}$ to $\frac{5}{8}$ in.) diameter.

In July 1901, Baum built a new washer at the Emscher pit of the Kölner Bergwerks-Verein, in which an attempt was made to wash coal without the preliminary division into a number of fractions for separate washing. Double wash-boxes were built, each consisting of two compartments, on either side of a central compressed air compartment, as shown in Fig. 37 ; Fig. 36 is the longitudinal section of the same wash-box. Each compartment was 4 metres (13 ft. $1\frac{1}{2}$ in) long, and 1·2 metres (3 ft $11\frac{1}{4}$ in.) wide, so that the total length of the box was 8 metres (26 ft. 3 in.), and the total available sieve area of each box was 17·28 sq. m. (186 sq. ft.). In each compartment of the wash-box there was a refuse-removal valve consisting of an arrangement of double gates controlled by levers, a device similar to that used in the earlier type of Baum washer. In the first compartment of each washing section, the refuse-removal valve was fitted at the end of the box where the raw coal was admitted, and was underneath the feed shoot. In the second compartment, the refuse-removal valve was fitted at the end of the box from which the washed coal was removed, and was placed under the washed-coal overflow weir. The position of the two gates of

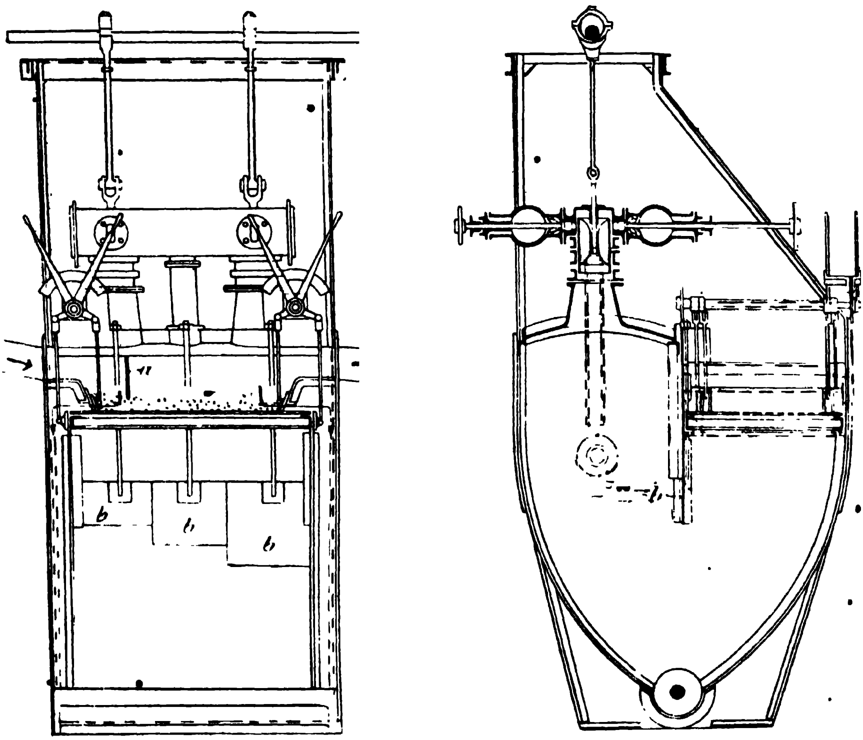
each valve above the level of the fixed sieve was such that the refuse could pass underneath the first gate and overflow over the second one. The refuse after passing the double valves settled to the



FIGS. 36 AND 37.—Longitudinal and Cross Section of First Baum Wash-box for Unsized Coal (1901)

bottom of the wash-box, being protected from the water pulsations in the main portion of the wash-box by suitable baffle plates. By setting the first refuse-removal valve at the feed end of the box, the heaviest dirt, which settled on the sieve as soon as it entered the

wash-box, was removed before it had traversed much of the length of the box. Consequently most of the sieve area was available for the settling of the lighter dirt. The finest sizes of dirt, through 8 mm. ($\frac{5}{16}$ in.), passed through the perforations of the sieve and settled to the bottom of the wash-box, where they were conveyed by Archimedean screws to bucket elevators set at each end. In the first compartment of the wash-box a plate, *a*, Fig. 36, was fixed vertically to spread the coal over the whole width of the box, and



FIGS. 38 AND 39.—Longitudinal and Cross Section of Later Baum Wash-box for Unsized Coal.

so aid the settling of the heavier dirt on the first portion of the sieve. Adjustable plates, *b*, Figs. 36 and 37, aided the control of the water pulsations in the wash-box.

In this new type of wash-box the direction of travel of the coal was parallel to the division plate between the air and the sieve compartments, and not at right-angles to it. It will be remembered that Lührig had used this direction of travel through his fine-coal feldspar washer, a method afterwards adopted in Coppée and other feldspar washers. It was, however, only applied to coal of size less than $\frac{3}{8}$ in. for capacities of 4 tons per hour. Baum used this direction

of travel for coal up to, say, 2 in. size for quantities of 50 tons per hour. By this important alteration, the horizontal length of travel of the coal through each washing division was 8 metres (26 ft. 3 in.) instead of being about 1 metre (3 ft. 3½ in.), as was the case in the earlier type of washer. Thus the total sieve area employed in this double wash-box was about the same as the total area employed in the earlier type of washer of the same hourly capacity, but in the earlier type it was divided between, say, eight wash-boxes, each of which required separate control, instead of, as in the new type, being confined to one large box.

Practical experience with the new washer showed that the coal treated was sufficiently cleaned after traversing the first compartment, and, in the next washer built, the length of the box was reduced to 6 metres (19 ft. 8½ in.) This newer wash-box was still built with a washing division on each side of the central air compartment, and the breadth of the sieve was the same as was previously employed. A third refuse-removal valve was, however, placed in the middle of the length of the wash-box. The total sieve area available was 12·96 sq. m. (139·5 sq. ft.). The heavy dirt was removed at the first refuse-removal port, the intergrown coal and shale at the middle port, and the light shale at the third port.

This new wash-box could treat the whole of the coal in one washing division 6 metres long by 1·2 metres broad (19 ft. 8½ in. by 3 ft. 11¼ in.), with an available sieve area of 6·48 sq. m. (69·7 sq. ft.). The building of double wash-boxes was therefore discontinued and the design illustrated in Figs. 38 and 39 was adopted. Apart from the use of a single washing division and the reduction in the length of the box, the design employed was similar to that of the earlier types. The partition plates, *b*, were, however, not of uniform depth in the whole length of a box, but were " stepped," the smallest plate being employed at the end of the box where the coal was admitted. By this means the water pulsations could be modified in the length of the wash-box, being strongest where the heavy dirt was deposited.

In some cases the use of a single wash-box did not give adequate cleaning of the smallest sizes of coal, and at Rhein Elbe III Colliery the fine coal was re-washed. At Zeche Concordia Schact IV a rewash-box was used for the crushed middlings and fine coal. The second wash-box was of similar design to the first, with the exception that no refuse-removal valve was fitted at the front end of the box.

Thus Baum, besides being able to wash coal before classifying, had been successful simultaneously in reducing the area of sieve necessary for efficient washing to about one-third of that which was used in the best washing practice of his day. In proving that it was not essential to subdivide the raw coal into a large number of fractions before washing it, he reduced the capital cost of the washery and greatly simplified its control. His improvements in washery design also enabled washers of larger capacity to be built, for although other washery designers had fed unsized coal to a wash-

box (as for example, in some of the older forms of "bash" washer), such washing processes were either inefficient or were only capable of dealing with small quantities of coal.

By the beginning of the twentieth century almost all washery designers had accepted the principles laid down by Rittinger and introduced by him into ore-dressing practice, and by Lührig into coal-washing practice. The principle chiefly involved was the specification of sizing limits for the separation of two types of material. The successful cleaning of unsized coal by Baum appeared to be in direct contravention of this principle. It was not, however, realised that (as pointed out in Chapter III), during jigging there is a period during which separation takes place according to density differences only, and independently of size. Because of the existence of this period it was found that, even though all coals of run-of-mine size could not be successfully washed in one box when large quantities (over 50 tons per hour) were treated, it was, in general, possible to wash all the coal from, say, 3½ in. to ⅜ in. size, with a sizing ratio of 9·3 to 1, in one box when the total quantity of coal was as great as 100 tons per hour. The coal of size less than ⅜ in. was frequently insufficiently cleaned by a single treatment in the one box and the employment of a second box for re-washing the product below ⅜ in. became necessary, but the coal from 3½ in. to ⅜ in. could always be satisfactorily cleaned in the one operation. The size ratio, assuming that cleaning could only be accomplished down to ⅜-in. size, was double the theoretical maximum ratio of Rittinger, and was about six times the ratio used in Lührig and similar types of jig washer.

The first experimental washer was built in 1901, and it was found to be so efficient in operation that, by 1905, over thirty-four Baum washers of the new type had been commissioned in Germany. The rights to build Baum washers in the British Empire were acquired by the firm of Simon-Carves, the British coke-oven construction company, who have modified and improved the design in the light of experience gained with its use since 1903. The capacity of the washeries has been gradually increased, as is shown in Table 59, in which the number and capacity of Baum washers built by Simon-Carves, in Great Britain, are recorded. Six-year periods are chosen, as they show the effect of the war years on washery construction.

Simon-Carves have also built in other countries sixteen Baum plants with a total hourly capacity of 1,205 tons.

Since 1922 a number of other washery construction firms have built Baum-type washers in Great Britain. These firms are Sherwood Hunter, Ltd., Penzance; the Coppée Company (Great Britain), Ltd., London; Horace Greaves & Co., Ltd., Derby; and Nortons (Tividale), Ltd., Tipton. These firms have built 26 plants of 2,785 tons per hour total capacity.

The designs of the Baum wash-box built by all washery constructors in Great Britain and Germany are the same in principle, and

differ only in detail. The Baum type of washer has proved to be the most popular of all washers in Great Britain, and is now preferred to all other types of jig washer. Most of the new washery constructors building Baum washers follow the standard Baum practice of washing before sizing, but the Coppée Company build Baum washers to wash either before or after sizing, in the latter method following their previous practice in Coppée nut, bean and pea washers. Fig. 40 is a stereoscopic photograph of a Coppée Baum washery, of 150 tons per hour capacity, in which the coal is sized before washing into nuts, $1\frac{1}{8}$ to $\frac{3}{4}$ in.; peas, $\frac{3}{4}$ to $\frac{3}{8}$ in.; and smalls, through $\frac{3}{8}$ in. The smalls fraction is washed in the right-hand box, and the nuts and peas are washed in the two compartments of the left-hand box.

The washery with the largest hourly capacity in Great Britain has recently been built at the Marine colliery of the Ebbw Vale Steel, Iron and Coal Co., Ltd. It was erected by Messrs. Horace Greaves and Co., Ltd., of Derby, and though rated at a capacity of 300 tons per hour, has achieved a throughput of 360 tons per hour. It comprises two separate units each of the Baum type, and each with a separate rewashing-box for the smaller sizes.

For the description of the working of a Baum washer, it will be sufficient to consider the Simon-Carves type of wash-box and washery, and, subsequently, to describe the modifications introduced by other constructors.

In Fig. 41 a view is given of a modern Baum washery of 160 tons per hour capacity, built by Messrs. Simon-Carves. Fig. 42 is a plan of a similar Baum washery, the section being taken at the level of the top of the wash-boxes. Fig. 43 shows a longitudinal section through the same washery in front of the wash-boxes. Fig. 44 is a cross-section, showing the screening plant and hoppers.

The raw coal is unloaded from wagons into an underground hopper, from which it passes by shoots on to one or other of two revolving feed tables. These feed it at a uniform rate into the boot of an elevator, 3, which carries the coal to the top of the washery

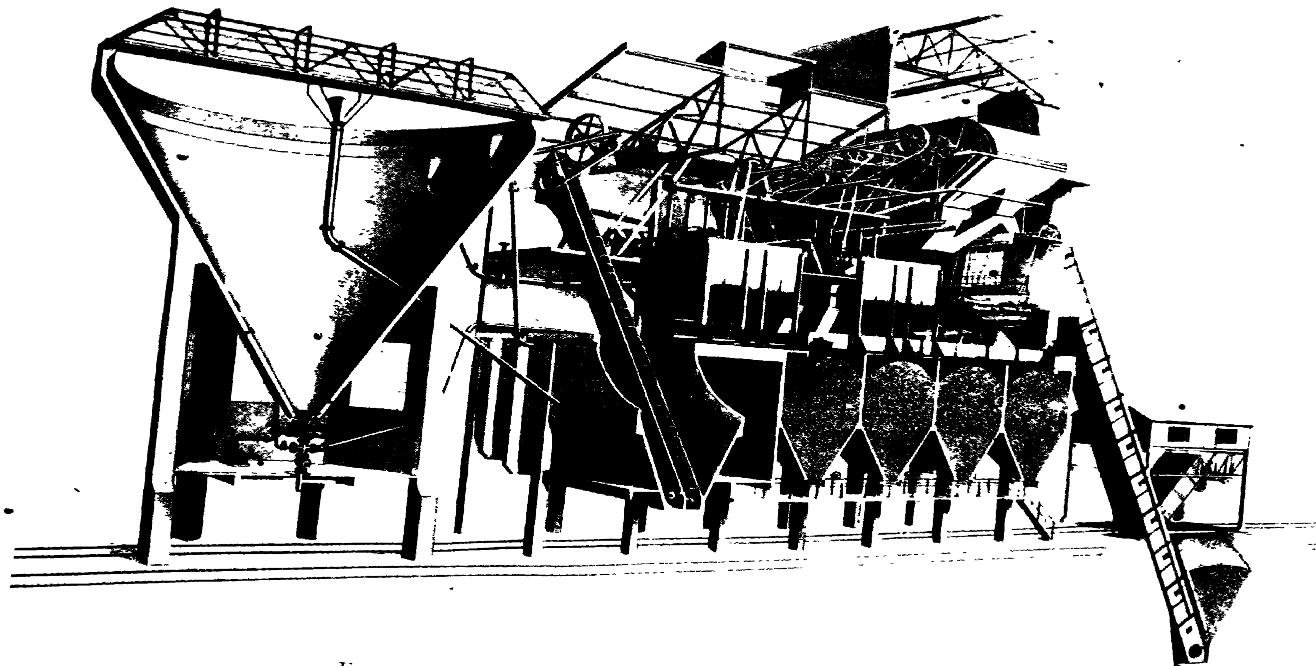


FIG. 49.—Stereoscopic Diagram of Coppée Baum Washery.

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FIG. 41.—External View of Simon-Carves Baum Washery.

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building. Here the coal passes by a shoot directly into the first wash-box, 4, Figs. 42 and 43, in which the larger-sized dirt, as well as a portion of the smaller dirt, is removed. Some of the dirt settles through the bed in the first compartment of the wash-box and passes through the screen to the bottom of the box, where a screw conveyor carries it along to the dirt elevator, 5. The heavier dirt which does not pass through the sieve, also passes continuously into

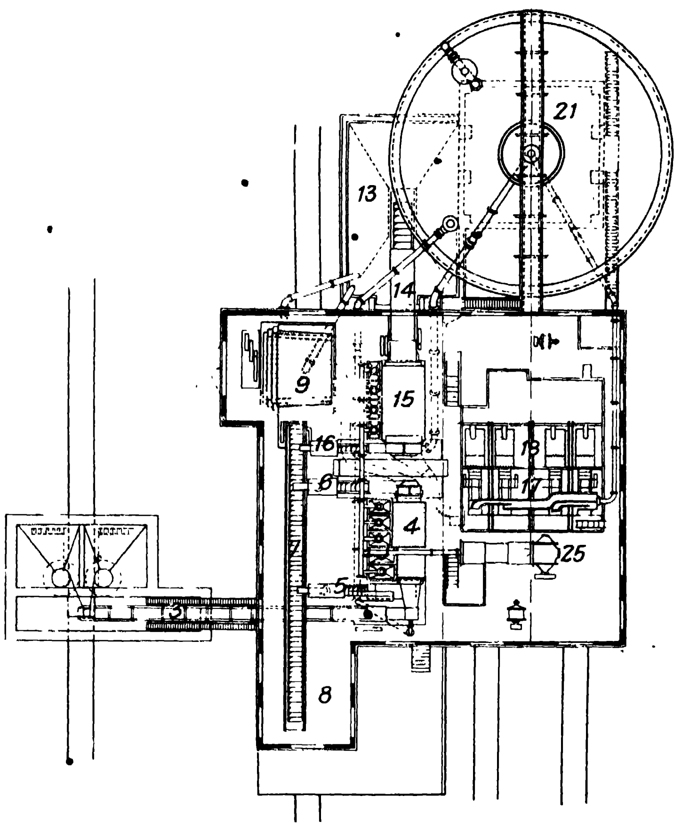

FIG. 42.—Plan of Baum Washery (Simon-Carves).

the refuse elevator, 5, through a refuse gate at the feed end of the box. The dirt is removed in a similar manner from the second compartment of the wash-box by means of a screw conveyor and a dirt elevator, 6. The washed coal leaving the first wash-box is freed from almost all the large dirt, but still retains the smaller dirt particles.

The whole of the washed product from this wash-box is therefore taken by means of a launder (shown by dotted lines in Fig. 42) from the outlet of the first wash-box, 4, to the revolving "trommel" screen, 9 (Fig. 42). This consists of a series of concentric

cylindrical sieves of different mesh, by means of which the coal may be divided into suitable sizes, for example, "nuts," "beans," "peas," and "fines." The "nuts" and "beans" are washed over drainage screens into bunkers, which are provided with spiral shoots to obviate the breakage of the coal as it is loaded into the hoppers (11, Fig. 44). The "peas" are also passed over a drainage screen into a bunker, which, however, is not provided with a spiral shoot.

The fine coal, which is that through ⅜-in. mesh, is usually only imperfectly cleaned in a washer dealing with 160 tons per hour, and is re-washed. By running the washed coal from the first wash-box to the "trommel" screen there is a loss of head, and in order to

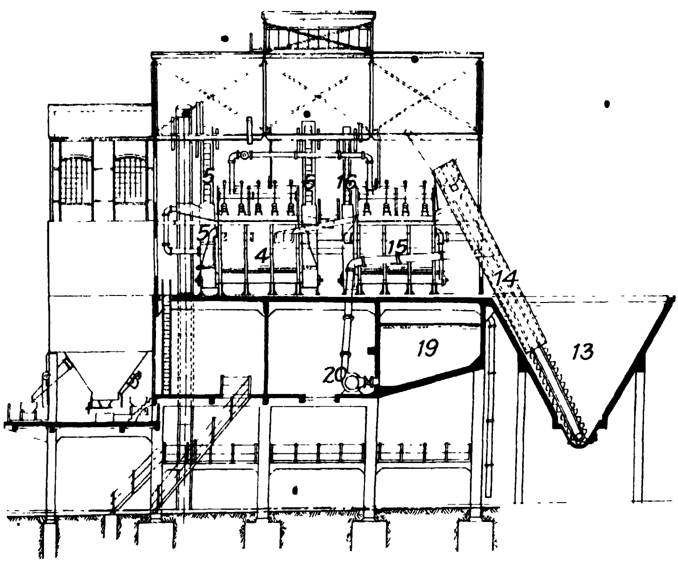

FIG. 43.—Sectional Elevation of Baum Washery (Simon-Carves).

recover this, and to have the two wash-boxes on the same level, use is made of an elevator. The fine coal flows in a stream of water to the fine-coal sump, 13, from which it is raised by the elevator, 14, to the fine-coal re-wash box, 15. The small dirt is removed from this box by a screw conveyor and elevator, 16. The three dirt elevators, 5, 6, and 16, drain the refuse and discharge it into a scraper conveyor, 7, which feeds the dirt bunker, 8.

The fine washed coal overflows from the wash-box, 15, to drainage screens, 17, and shaking screens, 18, from which it drops on to a conveyor which distributes the coal into bunkers, 24 (Fig. 44). The water drained from the fine coal on the drainage sieves passes into the fine-coal sump, 13. The water in circulation from both wash-boxes is therefore collected in this sump and overflows into the pump basin, 19. From here it is taken by the main washery pump,

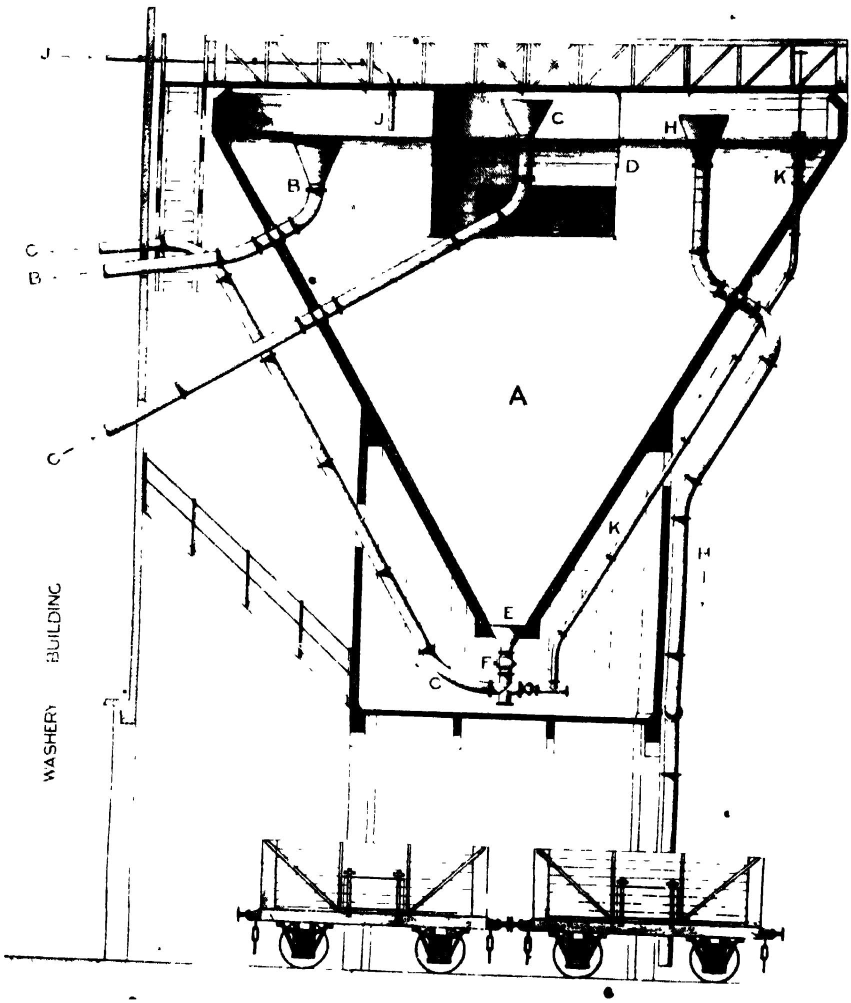

FIG. 45 —Elevated Conical Settling Tank (Simon-Carves).

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20, and discharged into the conical settling tank, 21, through the pipe, which can be seen in the centre of the settling tank. The air compressor and pressure-compensating reservoir, 25, supply the air through pipe lines to the wash-boxes.

The elevated conical settling tank is shown in vertical section in Fig. 45. The water containing the slurry pumped from the sump, 19 (Fig. 43), passes through the pipe, C, to the centre of the tank, A, into which it overflows from an inverted cone-shaped mouthpiece. This is surrounded by a steel curtain, D, which is designed to prevent

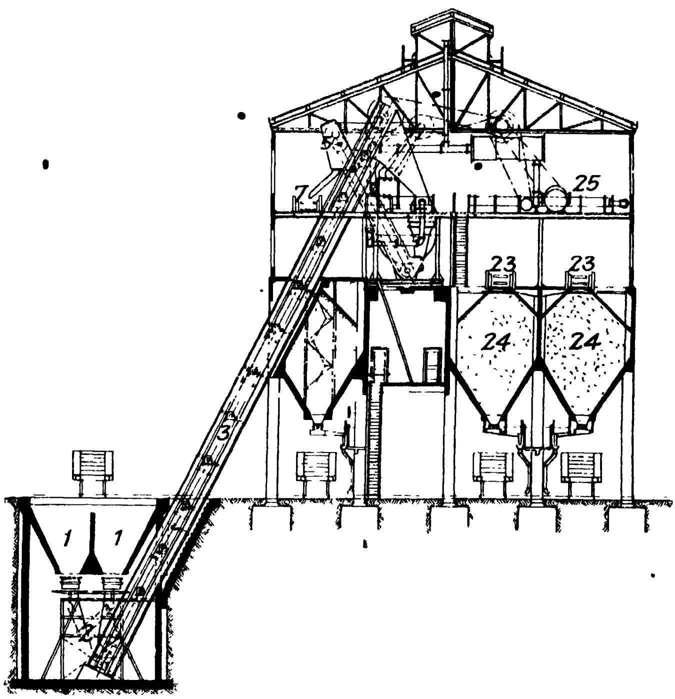

FIG. 44.—Cross Section of Baum Washery (Simon-Carves).

the mixing of the slurry with the clear water at the top of the settling tank outside the curtain. The slurry settles to the bottom of the tank and is removed by gravity through the outlet, E, and the cock, F, to the pipe, G, which returns it to the washery building in a concentrated form. The pipe carrying the concentrated slurry to the washery may be seen at the right-hand side of Fig. 42. In order to govern the consistency of the slurry, some water is admitted through the pipe, K. The slurry is added to the fine coal as it passes over the shaking screens before being distributed to the bunkers.

Referring again to Fig. 45, an overflow pipe, H, and a fresh-water supply pipe, J, are also illustrated. The tank is set at such a

height that clear water may readily flow by gravity through the pipe, B, to the wash-boxes. Sufficient head is also available to carry the slurry to the drainage sieves.

A sectional elevation of a Simon-Carves-Baum wash-box is given in Fig. 46; Fig. 47 is a cross-section of the same box, and Fig. 48 is a view of a wash-box from above, in which the air valves and the air mains may readily be distinguished. In Fig. 46 the coal enters the box at F. The fixed sieve, K, of the first compartment, slopes downwards towards the refuse port, G. The latter may be covered partly or wholly by a refuse gate, connected to spindles and a crank arm, worked by a handwheel (Fig. 47). A plate is fitted vertically at the entrance to the wash-box so that the feed coal is uniformly

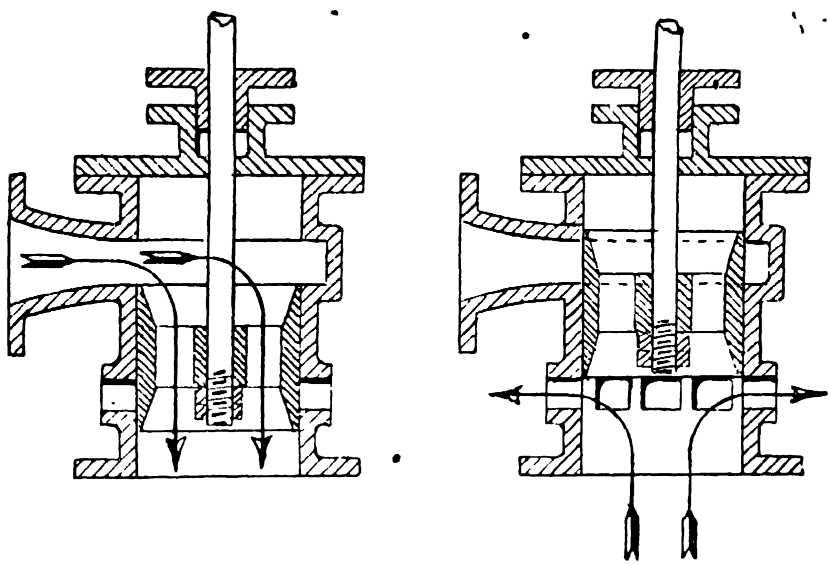

FIG. 49.—Baum Air-Valve. Diagrammatic.

distributed and is forced under the surface of the water to wet it thoroughly. The formation of aggregates of fine coal particles, which could float through the wash-box, is thereby avoided. R Fig. 46, is the fixed sieve of the second compartment, of which O is the refuse port. Between the two compartments, E and M. there is a weir to maintain the bed on the sieves whilst allowing the lighter coal to overflow from the first compartment, E, to the second, M. At the end of the compartment, M, is a second weir over which the washed coal overflows at N.

The bulk of the dirt, and more especially the larger dirt, is deposited in compartment E, and the inclination of the sieve, K, enables it to be removed rapidly. The rate of removal of the dirt is controlled by adjustment of the refuse gate over the port, G, through which the dirt passes, through the channel, H, to the boot of the

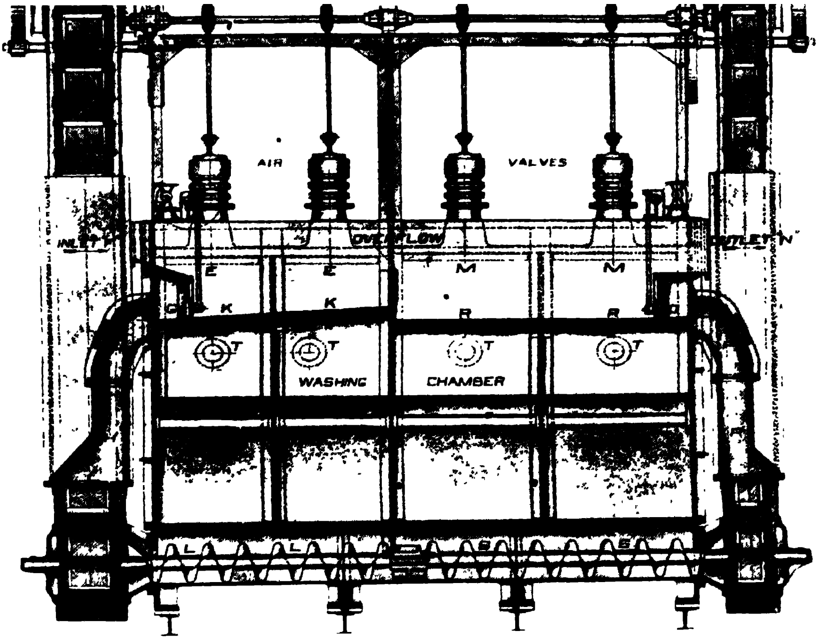

FIG. 46.—Sectional Elevation of Baum Wash-box (Simon-Carves).

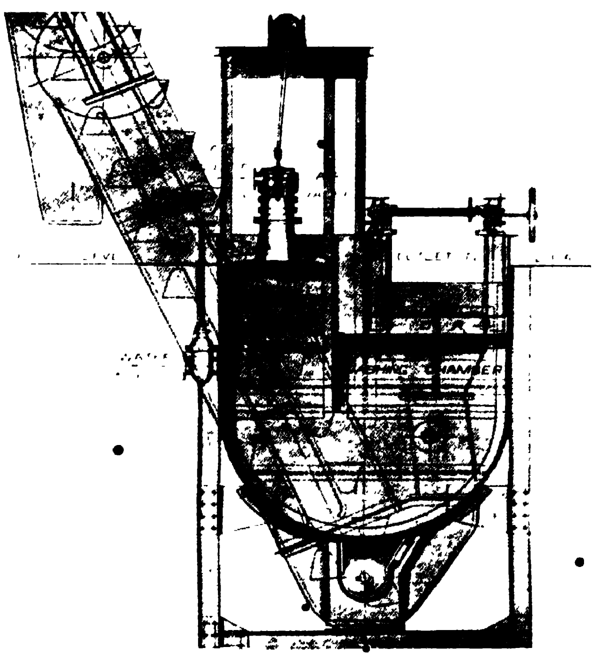

FIG. 47.—Cross-section of Baum Wash-box (Simon-Carves).

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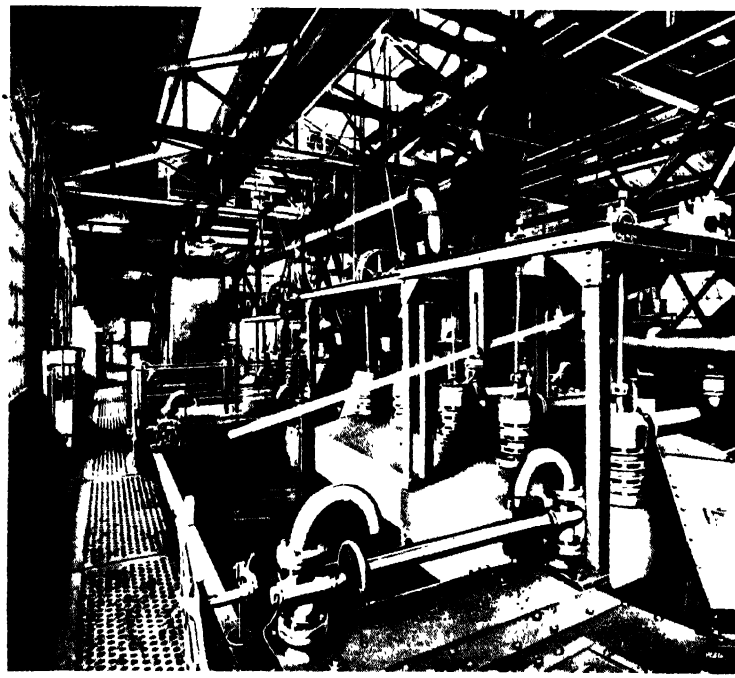

FIG. 48. View of Baum Wash box (Simon-Carves)

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elevator, J. Any dirt which settles through the bed and is small enough to pass through the perforations of the screen, settles to the bottom of the box, whence it is carried by the screw conveyor, L, to the elevator, J. The refuse is removed from the second compartment of the wash-box in a similar manner, by means of the refuse port, O, the channel, P, the screw conveyor, S, and the dirt elevator, Q. Fresh water enters the wash-box at four points, T, Figs 46 and 47. The countershaft and air valves of the wash-box are also clearly shown in Figs. 46 and 47.

From Fig. 47 it may be seen that each wash-box is divided into two portions, namely, the air chamber and the washing chamber. Between these two chambers there is a division plate, extending down below the level of the screen. This extended division plate prevents the disturbance of the upward water currents in the washing compartments by the eddies and downward currents in the compartment under the air chamber.

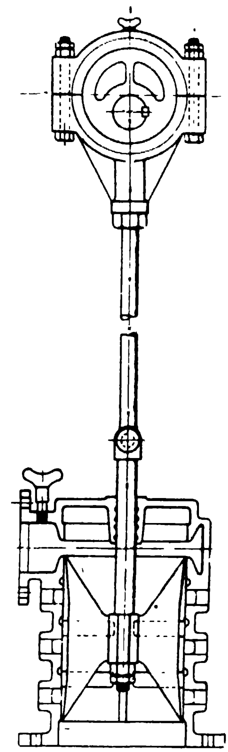

FIG. 50.—Baum Air-Valve : Later Type.

Fig. 49 is a sectional view of an air valve. The air valve consists essentially of a cylinder with inlet and outlet ports, and a cylindrical slide valve which moves vertically in the cylinder and is connected by a spindle to a cam shaft. The inlet port is connected with an annular cavity in the air cylinder, which permits the air to enter from the whole perimeter of the cylinder. The exit ports are openings spaced at equal distances around the perimeter of the lower end of the air cylinder. When the slide valve is near the end of the downstroke, the exit ports are closed and the compressed air from the blower passes through openings between the sleeve of the valve and the spindle and enters the air chamber of the wash-box. The level of the water in the air chamber is depressed with a consequent raising of the level of the water in the washing chamber. On the completion of the down stroke, the rotation of the camshaft causes the slide valve to rise; the inlet port is then closed and the exit ports are uncovered. The air pressure is released and the level of the water rises in the air compartment. This movement is repeated periodically so that a series of upward water currents are produced in the wash chamber.

Fig. 50 illustrates a more recent type of air valve in which three belts of exit ports are used.

The harmful effects of a downward current through the bed of the material to be washed have already been described. The means

adopted in the Baum washer for minimising the current of water flowing downwards through the bed are twofold. Firstly, the release of the air pressure occupies a longer time than that necessary for its production, and the velocity of the downward currents is proportionally less than the velocity of the upward currents. Secondly, the differentiation between the speeds of the upward and downward water currents so obtained is accentuated by the addition of water to the air chambers of the wash-boxes at the points, T, Figs 46 and 47.

The water supplied at T reaches the wash-box under a constant head of pressure from the elevated tank, through a cock, which, after once being set with the correct opening, remains open throughout the washing process. During the downstroke of the slide valve in the air cylinder, the pressure of the air, together with the pressure of the supply water, combine in forcing water from the air compartment into the washing compartment and in producing an upward current through the bed of material to be cleaned. During the upstroke of the slide valve the air pressure is released and the difference in pressure between the two compartments is due solely to their water levels. To equalise the levels water flows downwards through the bed. Because of the supply of water under a head of pressure at the base of the air compartment, however, the equalisation of pressure is retarded by water flowing into the air compartment from the elevated tank. By this arrangement, therefore, the water supply acts with the air pressure in producing an upward current, and limits the production of a downward current when the air pressure is released, so avoiding the harmful effects of " suction."

A single Baum wash-box is capable of washing up to about 50 tons per hour of unsized coal from, say, 2½ in. downward, and the smaller sizes of the washed coal (through, say, a ⅝-in. mesh screen) will be suitable for coke manufacture. For example, the following results were obtained in washing a typical South Yorkshire coal during the year 1924–25. The coal reaching the washer contained all sizes from 2½ in. downward. The portion of it below ⅝ in., which comprised 65 per cent. of the total, contained 18·01 per cent. of ash; after washing, the portion of the washed coal below ⅝ in. contained 7·15 per cent. of ash. During the process of washing, the loss of coal sustained was :—

Coal in washery dirt—	
(a) Large dirt	2·99 per cent.
(b) Small dirt	3·98 ,,
Loss of coal (expressed as a percentage of the total coal fed) .	0·72 ,,

In this case, therefore, 50 tons of unsized coal were washed per hour in a single box, and the ash content of the 0 to ⅝-in. coal was reduced by three-fifths with a total loss of coal of well under 1 per cent.

When, however, a capacity of 75 tons per hour of run-of-mine coal is required it is desirable to have two wash-boxes if the fine coal is to be used for coke manufacture. When washing run-of-mine coal in washers of small capacity, the coarser material keeps the bed in a condition sufficiently " open " to allow good separation of the coal from the shale. If only coal less than, say, ⅝-in. size is treated, the bed tends to set compactly and the washing results may not be so good despite the addition of a rewash box Thus the practice of washing unsized run-of-mine coal has the advantage of keeping the bed from setting compactly, and better washing results can frequently be obtained than by washing the same coal in a jig washer for which the coal is classified into several sizes before washing. In one Simon-Carves Baum washery known to the authors, only one wash-box is required for a capacity of 130 tons per hour, but this is exceptional, and in this instance it is only the quality of the coal that makes it possible.

The dimensions of the boxes required for a washery of 75 tons per hour capacity, washing a normal run-of-mine coal for coking purposes, are : First wash-box, length 15 ft. (4·6 metres), width 8 ft. (2·45 metres) ; rewash box, length 13 ft. (4.0 metres), width 8 ft. (2·45 metres), giving a total washing area of 224 sq. ft. (20·8 sq. m.), which is equivalent to 3 sq. ft. per ton of coal washed per hour.

For a washery of 150 tons per hour capacity the average dimensions of the wash-boxes are : First wash-box, length 16 ft. 6 in. (5·05 metres), width 11 ft. 6 in. (3·05 metres) ; rewash box, length 16 ft. (4·85 metres), width 11 ft. 6 in. (3·5 metres), that is a total washing area of 374 sq. ft. (34·7 sq. m.), equivalent to 2·5 sq. ft. per ton of coal washed per hour. The dimensions are altered to suit the quality of the coal dealt with. It will be observed, by comparison of these figures with those for the earliest types of Baum jig used for unsized coal, that the length of the wash-boxes has been reduced somewhat but that the width has been doubled for a 75 tons per hour unit, and trebled for a 150 tons per hour unit This is the main factor which has allowed the increase in capacity from 50 to about 150 tons per hour. It is generally accepted that this is the limit of capacity for one unit.

The size of mesh used in the fixed sieves and the number of strokes of the air-valve cylinder made per minute are also varied to suit the quality of coal employed. In a normal plant the size of the mesh used in the first wash-box is ½ in., and in the rewash-box, $\frac{1}{16}$ in. The use of these sizes of openings in the sieves allows the removal of the smaller sizes of dirt from the bed and enables it to be kept in a fairly " open " condition. The average number of pulsations made per minute is 55, the number of pulsations being the same in each wash-box. The depth of water displacement can be varied by throttling the air valve. The air pressures used vary from 1½ to 2½ lb. per sq. in. By closing the air valve, the water pulsations can be stopped in the event of a sudden cessation of the coal supply.

There are two features of the Sherwood Hunter type of Baum washery requiring special description. The first is the arrangement for slurry recovery and water clarification, and the second, an automatic control of the fine-dirt discharge. When the valve at the base of the conical settling tank is opened, the pressure of the supernatant water forces much of the deposit out of the tank. The water, however, may not depress the level of the solid deposit uniformly, but force a channel through the centre. When this happens the discharge from the valve consists mainly of water, carrying with it a relatively small number of solid particles washed from the sides of the channel. The device fitted by Messrs. Sherwood Hunter prevents the formation of this channel and causes a considerably higher proportion of solids to be discharged with the water from the settling tank. In consequence the deposit can be removed with less disturbance and loss of the water from the tank.

The fine refuse which collects at the bottom of the wash-box is removed by a screw conveyor, as in the Simon-Carves wash-box. Occasionally the screw conveyor may clog and fail to function correctly. Unless the washery foreman notices the absence of fine dirt from the buckets of the refuse elevator, the fact that the conveyor is not working properly will escape notice, with the result that the efficiency of washing suffers until the week-end, when the wash-box must be emptied and cleaned out. In the Sherwood Hunter washer the dirt elevator and the fine dirt screw-conveyor are operated by the same drive so that the dirt conveyor is directly driven.

The direct drive is illustrated in Fig. 51, in which sectional elevations of the primary and the rewash boxes are given. It was found by careful testing that the screw conveyor absorbed more power than the dirt elevators, and, therefore, a direct drive for the screw conveyor and a belt drive for the elevators was considered to be more rational. With this arrangement the screw conveyor will only cease to function if it breaks, and slip of the drive on the elevators is easily noticed.

It will be observed that Sherwood Hunter increases the length of the primary wash-box and uses three air valves in the second compartment instead of two. The washed coal is then screened and the size through $\frac{1}{8}$ in. only is rewashed in a second box, which is fitted with a wedge-wire screen.

In the Coppée Baum-type washer, illustrated in sectional elevation and cross-section in Fig. 52, the raw coal enters through A and passes through the two compartments, B and E, the washed coal overflowing at J. The coarse dirt passes under the slide, M, through the conduits, C and F, to the boots of the elevators, D and G. The smaller dirt passes through the screen plates and is removed by the screw conveyors, H and I, to the elevators, D and G. The air pulsations are obtained in the compartment, K, by a rotary valve, L,

by means of which the quantity of air admitted and the rate of admission and release can be adjusted. The air-release cocks, N, are used to govern the evacuation of the refuse in conjunction with

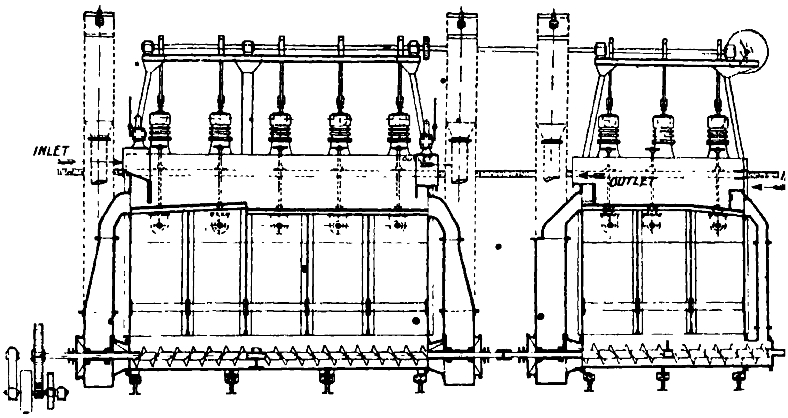

FIG. 51.—Longitudinal Section through Baum Wash-boxes (Sherwood Hunter).

the adjustment of the gate valve, M. The rewash box is similar in general design to the primary wash-box except that the ratio of the widths of the sieve and the air compartments is made 16 to 7 instead of 12 to 11 as in the primary wash-box.

Nortons (Tividale) in their latest plants use a different form of

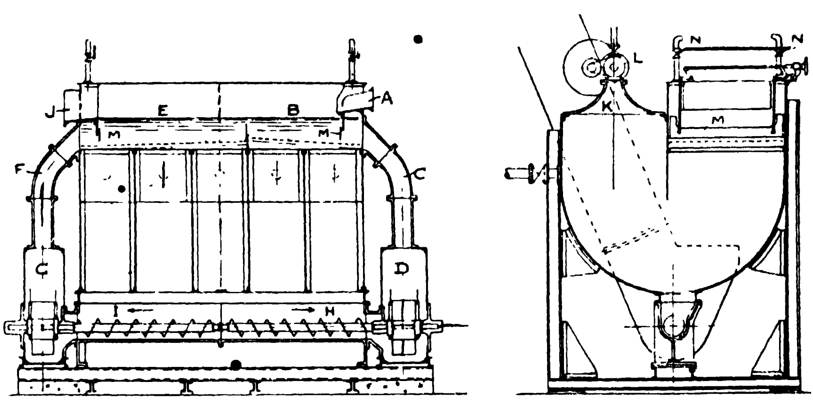

FIG. 52.—Longitudinal and Cross Section through Baum Wash-box (Coppée).

refuse-discharge gate, actuated automatically by a float immersed in the washing bed.

With the small amount of middlings usually found in British coals it is not customary to collect a separate middlings fraction. If it is desired to do so, however, all the large heavy dirt is extracted

in the first compartment of the primary wash-box, and the middlings are recovered as the reject in the second compartment and collected in the second elevator. Fine dirt particles which are deposited in the second compartment, and pass through the screen, are collected by the lower dirt worm and pass to the first (dirt) elevator. If middlings are recovered they are crushed and returned to the primary box for rewashing. If it were desired to make a separate middlings fraction the uncrushed middlings might be rewashed in a separate box.

The control of a Baum washer is based upon the thickness of the bed, and its " feel " (as described for the Humboldt washer, p. 150) and, in case of variation from the conditions for which the wash-boxes have been set, the main water-valve may be adjusted. The air pressure is also controlled by a single valve. The provision of dirt elevators at the ends of the wash-boxes enables the washery man to inspect the refuse almost immediately after its rejection, and to make any necessary adjustments of the dirt slides or of the water and air valves. At each end of the wash-box, the casting over the refuse-removal port " blinds " part of the screen, and to prevent an air lock which would interfere with the discharge of the dirt, air-release pipes are carried upwards from the casting, and are provided with cocks, as shown in Figs. 46 and 47. Should the feed stop altogether, the air-release valve is opened and the main water valve closed. On restarting, the air-release valve is closed and the water valve opened.

A standard feature of an older type of Baum washery was a drainage band conveyor for dewatering the fine coal. It consisted of a series of boxes made of perforated plate, the boxes being hinged one to another by links at the middle of the base plates. The drainage-band was driven by rectangular drum heads, and was suitably supported on rollers. A more complete description is given in Chapter XXV, and it is sufficient to say at this stage that the drainage band was not entirely satisfactory. In 1914 it was replaced by a simpler and more efficient arrangement of double shaking screens, which occupied much less ground space and absorbed less power. The drainage band built in Baum washeries prior to 1914 is being replaced in numerous cases by the dewatering shaking screens with much more satisfactory results.

The total bulk of water in a Simon-Carves Baum washer of 75 tons per hour capacity is 120,000 gallons, and the whole of this is circulated in one hour. In a plant of 150 tons per hour capacity, the total bulk of water is 230,000 gallons, of which 200,000 gallons is circulated per hour. In a 75 tons per hour plant the power requirements are 150 h.p., and in the 150 tons per hour plant, 300 h.p., or 2 h.p. per ton hour.

Some results of washing in Baum washers of different sizes are recorded in Table 62A, and, in Table 62B, the results of rewashing coal less than $\frac{1}{4}$ in. and less than $\frac{1}{8}$ in. size are given.

TABLE 62A.—RESULTS OF WASHING IN BAUM WASHERS

Size of Washer. (tons per hour).	Per cent. of Coal in Refuse. Elevators No. 1.	No. 2.	No. 3.	S.G. Chosen	Per cent. of Dirt in Washed Coal. Large Nuts.	Small Nuts.	Fines.	S.G. Chosen.	Per cent. Ash in Washed Coal. Large Nuts.	Small Nuts.	Fines	Remarks
40	1·6	1·0	—	1·40	—	—	—	—	—	—	6·3	4 month average.
50	3·0	4·0	—	—	—	—	—	—	—	—	7·1	1 year average.
100	2·1	0·8	2·1	1·35	0·5	2·3	4·0	1·60	6·6	7·1	7·7	6 day average.
100	2·7	2·3	2·9	1·45		6·4		1·45	—	—	—	1 month average.
100	1·5	2·9	5·2	1·40	—	—	7·4	1·40	—	—	6·2	1 month average.
100	2·8	0·2	1·0	1·48	—	—	4·7	1·48	—	—	8·6	Single test.
120		0·48		1·49	1·5	1·7	5·0	1·49	3·1	3·3	5·6	1 month average
130		2·9		1·40	—	—	6·4	1·40	—	—	6·0	3 month average.

TABLE 62B.—REWASHING COAL THROUGH $\frac{1}{4}$ IN. AND THROUGH $\frac{1}{8}$ IN.

Size.	Raw Coal. Weight per cent	Raw Coal. Dirt per cent.	Washed Coal. After first box Weight per cent.	Washed Coal. After first box Dirt per cent.	Washed Coal. After second box Weight per cent.	Washed Coal. After second box Dirt per cent.
$>\frac{1}{4}$ in. .	0·5	—	0·4	—	0·4	—
$\frac{1}{4}-\frac{1}{8}$ in. .	51·3	9·4	47·2	4·0	44·4	2·1
$\frac{1}{8}$-20 mesh .	31·4	9·2	25·5	8·1	38·0	3·7
$<$20 mesh .	16·8	N.D.*	26·9	N.D.*	17·2	N.D.*
Total .	100·0	—	100·0	—	100·0	—
$>\frac{1}{8}$ in. .	1·0	10·0	24·2	3·4	28·1	2·0
$\frac{1}{8}$-20 mesh .	55·1	11·7	53·6	4·1	62·9	2·2
$<$20 mesh .	43·9	7·9	22·2	16·3	9·0	3·7
Total .	100·0	10·0	100·0	6·6	100·0	2·3
Ash per cent.	—	10·9	—	7·8	—	4·9

* N.D.—Not determined.

The raw coal itself was fairly clean in the finer sizes, but the results show a substantial removal of the finer dirt. Sinnatt and Mitton (*Trans.* Inst. Min. Eng., 1923–24, 67, 497) record an example in which a coal less than $\frac{1}{4}$ in. size, with 80 per cent. through $\frac{1}{8}$ in., and 36·3 per cent. through 1 mm., was washed at the rate of 20 tons per hour in one Baum box, and the washed coal contained on an average 5·3 per cent. of ash.

CHAPTER IX

MISCELLANEOUS JIG WASHERS

THE types of jig washer that have been described in the preceding chapters include the Bérard, the Sievers, the Marsaut, the Hartz, the Lührig (and similar types, the Coppée, the Schüchter-

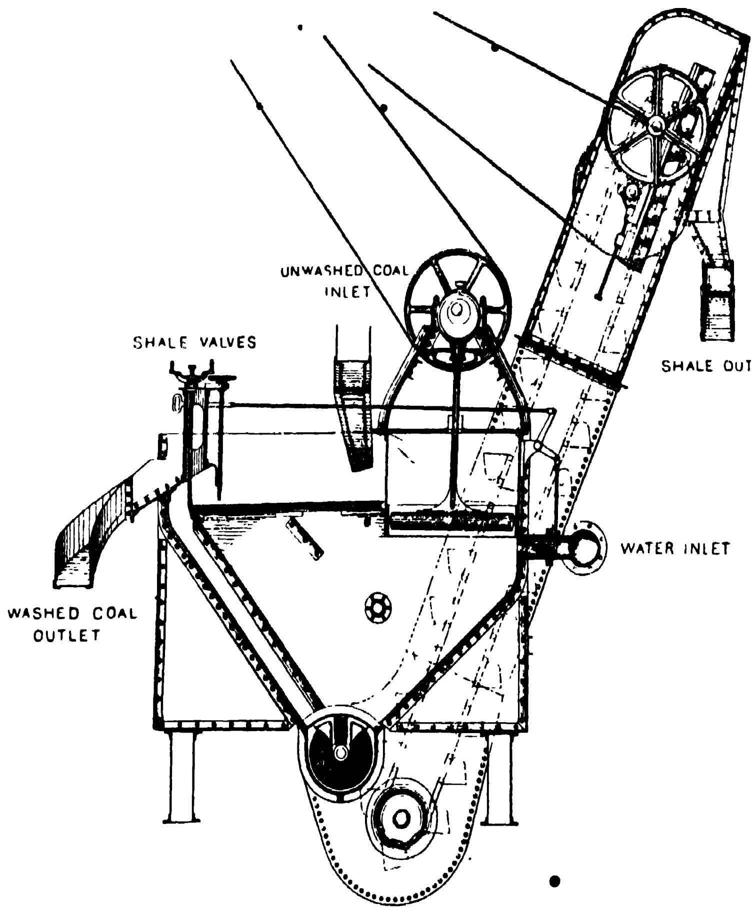

FIG. 53.—The Coppée Nut, Bean and Pea Washer.

mann and Kremer, the Barop, and the Sheppard), the Humboldt, and the Baum. Each of them is of considerable historical interest, since each marks a definite stage in the development of the modern jig washer.

A number of other types of jig washer are worthy of description because of the large numbers in which they have been built, although

they may be similar in principle to those described. Such are the Coppée, the Gröppel, the Howatt, the Elmore, and the Meguin.

The Coppée Washer.—As already described, the Coppée washer was developed as a modification of the Lührig jig. Coppée usually divided the raw coal into fewer fractions than did Lührig, for example, into fractions over $\frac{3}{4}$ in. (nuts), $\frac{3}{4}$ to $\frac{5}{8}$ in. (beans), $\frac{5}{8}$ to $\frac{3}{8}$ in. (peas), and through $\frac{3}{8}$ in. (fines). The Coppée washer for the larger sizes became known as the nut, bean and pea washer, the fines washer being a feldspar wash-box. The earlier form of nut-coal jig was similar to the Lührig nut-coal jig already illustrated; a later form is illustrated in Fig. 53 in longitudinal section.* The coal passed across the sieve, the washed coal overflowing into a shoot and the large dirt being removed below a refuse gate. The

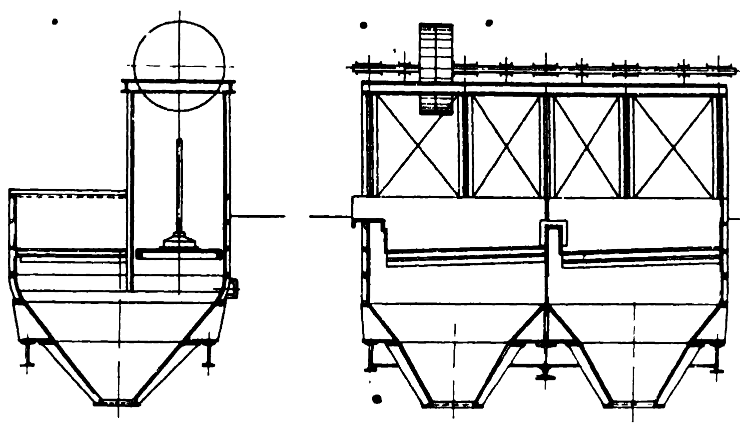

FIG. 54.—The Gröppel Nut-Coal Washer.

large dirt was separated from the water in the lower portion of the wash-box by a baffle plate until it was caught by a screw conveyor working in the bottom of the box. The finer dirt settled through the sieve to the screw conveyor and the total refuse was removed to an elevator. Water was admitted to the box below the plunger to modify the effect of the suction stroke.

The Coppée feldspar washer, like the Lührig feldspar washer, was a two-compartment "flow" jig. The later form of this washer was similar in many respects to the nut-coal jig except for the direction of travel of the raw coal through the jig, and the absence of dirt valves above the sieve. The dirt settled through the coarse feldspar particles, and through the sieve, to the bottom of the box, where it was run off through a valve in each compartment. Water was admitted above the plunger.

* It may be noted that the Coppée was not a "flow" but a "cross" jig, and the longitudinal section of the "cross" jig therefore corresponds to a cross-section of a "flow" jig.

The Gröppel Washer.—The Gröppel "flow" jig which, in 1905, replaced the Lührig "cross" jig in Germany, is similar in many respects to the Humboldt jig already described. In Fig. 54, sections of a modern Gröppel nut-coal jig are shown. The plungers are actuated by eccentrics. Each wash-box is made up of two compartments, the lower portions of which are pyramidal to collect the dirt which passes through the fixed sieve plate, and underneath the refuse-removal valves, to the boot of an elevator. This method avoids the use of a screw conveyor. In the fine-coal jig the design is similar, except that the ratio of the areas of the sieve and the plunger is increased, and the inclination of the fixed sieve is reduced.

Some of the details of Gröppel wash-boxes are recorded in Table 63.

TABLE 63.—DETAILS OF GRÖPPEL WASH-BOXES

	Nut coal.	Fine coal.	Re-wash.
Maximum dimensions of sieves	18′ 1″ × 5′ 11″	21′ 8″ × 7′ 3″	4′ 11″ × 10′ 10
Ratio of sieve and plunger areas	1·3 1·1	1 8–1·5	1·5
Dimensions of square sieve openings (mm.)	6- 13	4- 12	2–8
Number of strokes per min.	50 80	70 -100	70–90
Length of stroke (in.)	2–3¾	¾–2¼	¾–2¼
Washing capacity per sq. ft. of sieve area (tons per hour)	0·86–1·51	0·32–0·65	0·21–0·54

Fig. 55 is a view of a modern Gröppel washery built in 1922 at Zeche Sachsen, Hamm, Westphalia. The total capacity of the washery is 400 tons per hour, and includes two units each of 200 tons-per-hour capacity. The coal treated is through 80 mm. (3⅛ in.), of which 60 per cent. is less than 10 mm. (⅜ in.) size. The raw coal is graded before washing into two fractions, namely, 0 to 10 mm. and 10 to 80 mm. The nut coal (of 10 to 80 mm. size) is treated in a nut-coal jig and three products are produced, namely, clean nuts, middlings and large dirt (refuse). The fines (0 to 10 mm.) is subjected to an air blast to remove the material below 0·5 mm. ($\frac{1}{50}$ in.), and then passes through the fine-coal jigs, where three products are also recovered. The middlings from the nut-coal wash-box are crushed and rewashed in a separate unit. The middlings produced in the fine-coal jigs are not rewashed, but are mixed with the washed, crushed middlings from the rewash-box for boiler-firing purposes.

The architecture of this washery is worthy of note, although—in English eyes—it is somewhat massive for its purpose.

The Howatt Combined Jig.—The Howatt jig is the modern form of the Lührig washer built by Messrs. Coal and Ore Dressing Appliances, Ltd., in Great Britain. In the newer form, the design of the individual wash-boxes is essentially the same as in the older

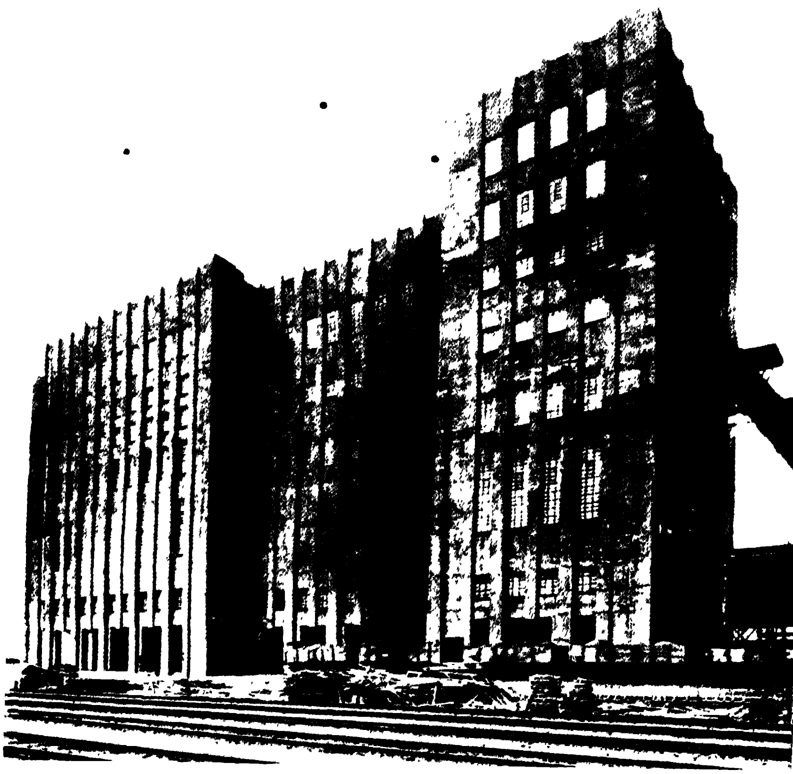

FIG. 55.—External View of a Modern Groppel Washery.

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Lührig washer, but the fine-coal feldspar wash-boxes and the nut-coal wash-boxes are arranged in separate rows with their plunger compartments adjacent and separated only by a small distance. In this space a dirt elevator operates, and is common to both rows

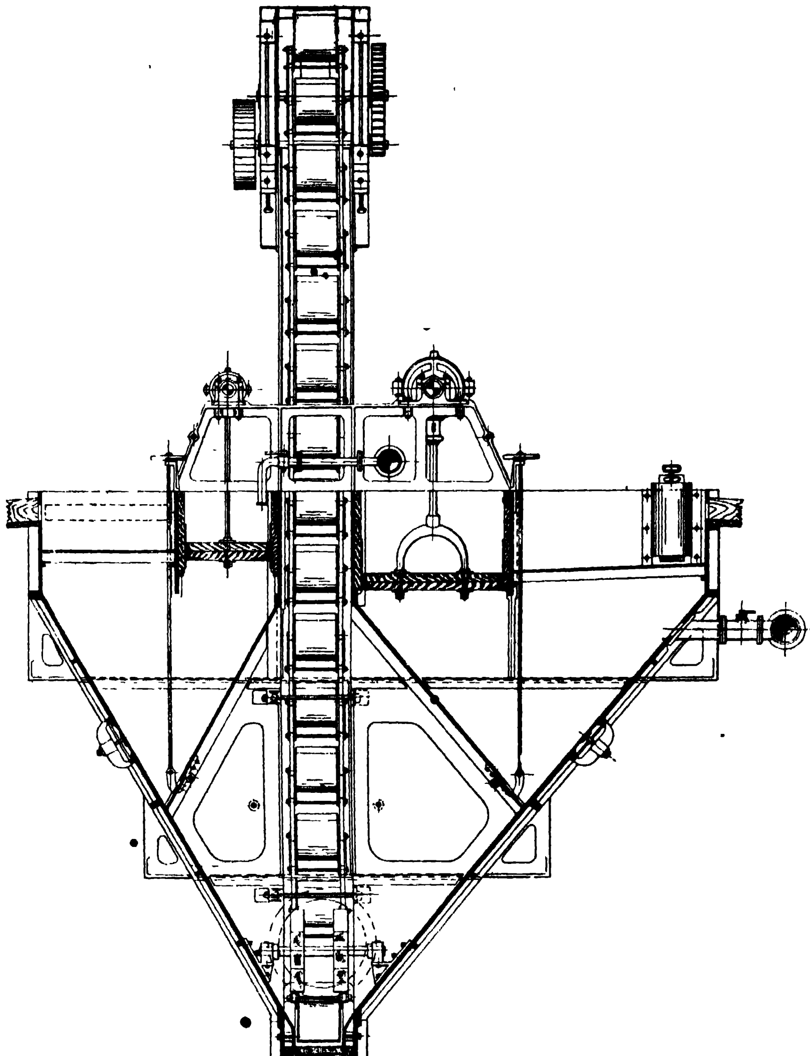

FIG. 56.—The Howatt Combined Jig.

of wash-boxes, as shown in Fig. 56. It will be observed that the outside outer sloping sides of the two rows of wash-boxes are extended until they meet to form a trough. Each wash-box communicates with this trough through openings on the inner sloping side of the box, the amount of opening being governed by a valve actuated

by a hand-wheel at the top of the box. The refuse from the fine-coal boxes passes through the feldspar and through the sieve over its whole area. It then settles to the refuse-discharge opening, and passes through this to the lower collecting trough. The dirt from the nut-coal jigs is discharged through a special cylindrical valve which Lührig first introduced. A cylinder extends through the total depth of the bed and almost touches the sieve. The dirt which settles on the sieve is forced up this cylinder by the pulsations. In two wash-boxes the dirt cylinders are placed at adjacent sides, between which a gap is left so that the refuse may overflow and settle to the refuse-collecting trough. One such valve is illustrated on the right-hand of Fig. 56. A scraper elevator working in the refuse-collecting trough removes the refuse from the washer and discharges it into a hopper. One elevator is therefore sufficient to remove the refuse from a combined jig of eight wash-boxes.

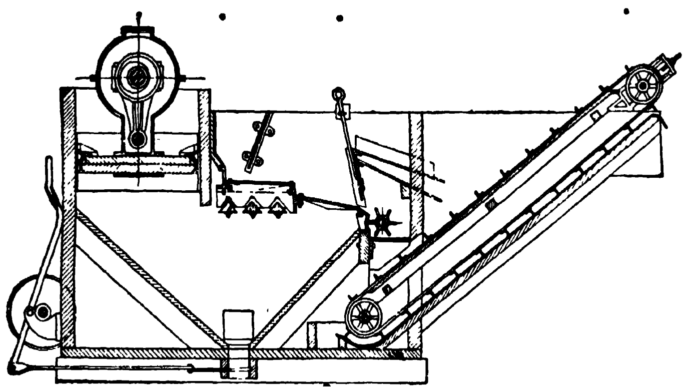

FIG. 57 —The Elmore Jig.

The Meguin Washer.—In a Meguin washer of 150 tons per hour capacity, which was recently erected at Anna I. Colliery, Kohlscheid, Aix-la-Chapelle, the raw coal is divided into two fractions, namely, 0 to 12 mm. ($\frac{1}{2}$ in.), and 12 to 80 mm. ($\frac{1}{2}$ to $3\frac{1}{8}$ in.). The larger size passes to a nut-coal wash-box, in which a plunger works below the sieve, and three products are produced, namely, clean coal, middlings and refuse. The fine coal, of 0 to $\frac{1}{2}$ in. size, is passed through a dust extractor, where the dust up to approximately 1 mm. ($\frac{1}{25}$ in.) is removed before delivery to the fine-coal wash-box, where three products are also recovered. The coarse middlings are crushed, mixed with the middlings from the fine-coal wash-box, and rewashed in a separate unit.

The Elmore Jig.—Over 250 Elmore jigs have been erected in the United States of America, but all are of low capacity. The washer is a double-compartment jig of the plunger type, the plunger being operated by eccentrics. The sieve in the fine-coal washer is inclined

downwards at a slight angle, but in the nut-coal washer it is divided into two sections, the back, or feed, half being horizontal and the front, or discharge, half being inclined slightly to facilitate the removal of the refuse. The washed coal is discharged over a weir on to a drainage screen, the refuse being removed by a scoop-wheel through a gate, the amount of opening of which is controlled by an adjustable plate (Fig. 57).

To obtain a current of uniform speed over the whole area of the sieve, oak beams of triangular section are placed underneath the back half of the sieve. On the downstroke of the plunger, on account of the shape of the hutch (which is triangular in vertical section), there is a tendency for the bulk of the water displaced to pass through the sieve near to the feed end. The oak beams impede the flow of water over the back half of the sieve and ensure a more uniform distribution than would otherwise be obtained.

The means adopted to prevent an appreciable downward current of water through the bed during the upstroke of the plunger consists of rubber flap-valves attached to the top edge of the plunger. During the downstroke the valves remain closed; on the upstroke the rubber flap-valves yield and prevent the tendency to form a vacuum under the plunger. By this means the displacement in the plunger chamber is made up largely by water passing downwards past the sides of the plunger instead of by an appreciable current of water through the bed.

The operation of the star-wheel, which withdraws the refuse from the sieve, is controlled by an automatic device. This consists of a floating piston enclosed within a float chamber. The float chamber is placed so that its base is 3 or 4 in. above the sieve (the thickness of the bed of refuse is about 4 in.), which brings it just below the top layer of refuse. On the downstroke of the plunger, the piston receives an impact from the refuse forced upwards against it, and the upward movement thereby imparted to the piston is transmitted through a connecting rod to a simple lever arrangement. The lever therefore acquires a reciprocating movement, during the jigging action, which it imparts, at the end opposite the connecting rod, to a pawl. The pawl engages in the teeth of a ratchet wheel which is attached to the rotating refuse-discharge valve (or star-wheel). At each upward movement of the bed of material on the sieve, the pawl is actuated and rotates the ratchet. During the upstroke of the plunger of the jig, the pawl slides over a tooth of the ratchet and engages with the next tooth, to rotate the ratchet again on the subsequent downstroke of the plunger. By this means the rate of discharge of the refuse is controlled automatically and the proper thickness of the bed is preserved. It is claimed that the device is simple and effective, and that it deals satisfactorily with a sudden increase in the amount of shale reaching the jig. If the amount of shale in the coal decreases, so that the thickness of the bed of refuse diminishes, the pawl is automatically lifted clear

of the ratchet until the proper thickness is restored. The whole arrangement would appear, however, to be rather more complicated than its purpose merits.

The length of the stroke required for the best operation of the Elmore jig varies from about $\frac{7}{8}$ in. for fine coal (half of which passes a $\frac{1}{8}$-in. screen) to $2\frac{1}{2}$ in. for nuts up to 2 in. size.

AMERICAN MOVABLE-SIEVE JIGS

Jigs with movable sieves are still used to a considerable extent in America, although their use in England has been confined to small plants only (the Greaves and the Hoyle). The Baum jig washer, which has proved so popular in Germany and in England, has not been erected in America. The types most popular—in addition to the Elmore jig—include three movable sieve jigs, namely, the Stewart, the Pittsburgh, and the American.

The Stewart Jig.—The Stewart was the first important jig with a movable sieve erected in America, and it acquires additional interest in that it was used extensively for the washing of unsized coal. The sieve on which the coal is washed forms the floor of a basket suspended by steel rods from a double eccentric mechanism. The raw coal is admitted at one end of the basket and the washed coal flows over a weir at the opposite end. The refuse is discharged through a gate, which is opened once during each reciprocating movement of the sieve. The width of the aperture at the gate during each movement of the sieve is controlled by the operation of a lever.

The differentiation between the speeds of the upward and downward currents of water through the bed of material to be washed is provided by a system of valves in the water-supply pipes, which prevent suction during the upward movement of the sieve. The valves remain closed whilst the basket moves downwards, thus forcing a current of water through the bed. During the upward movement, the valves are opened automatically by the tendency to form a vacuum under the sieve, and the inflowing water prevents a strong current of water passing downwards through the bed, the bulk of it coming from the fresh water supply.

The Stewart jig is made in single compartments, each of which has a capacity of 20 to 40 tons per hour of unsized coal. The cleaning effected may be illustrated by the following results relating to the cleaning of coal at the Sayreton Mines of the Republic Iron and Steel Company, and taken from Fulton ("Coke," Scranton, 1905, 120):—

	Specific Gravity		
	<1·37	1·37–1·56	>1·56
Raw coal	82·6	11·4	6·0
Washed coal	87·9	10·3	1·8
Refuse	3·8	18·2	78·9

The coal was run-of-mine, though the maximum size is not specified. The cleaning effected cannot be described as particularly efficient, for the ash content of the coke made from the washed coal was 14·1 per cent., whereas from the unwashed coal the ash content of the coke was 18·85 per cent. Further results are recorded by Garman, *Trans.* Can. Inst. Min. Met., 1922, 25, 446, as follows :—

	Total Sample. Ash per cent.	Floats.*		Sinks.*	
		Weight.	Ash.	Weight.	Ash.
Raw coal . .	22·8	56·0	9·4	44·0	43·4
Washed coal .	14·3	77·7	10·2	22·3	22·7
Refuse . .	67·0	2·5	16·0	97·5	68·2

* S.G. not given.

The Pittsburgh Jig.—The Pittsburgh jig is similar to the Stewart jig, but the basket composing the jigging sieve contains a false bottom provided with flap valves, to prevent a current of water passing downwards through the bed whilst the sieve is moved upwards.

A further modification is the substitution for the eccentric driving mechanism of a slotted lever and crank fitted with a hardened steel roller and brass bushes. During two-thirds of the revolution of the driving gears, the plunger is drawn upwards and the down-stroke occupies only one-third of the revolution. By this means the duration of the upstroke is made double that of the downstroke. This provision, together with the flap valves below the sieve, prevents an excessive downward current.

At a washery in Southern Illinois, U.S.A., the coal is crushed and screened prior to washing into three sizes, over 1 in., 1 to ¼ in., ¼ in. to 0. Each size is then washed separately in Pittsburgh jigs prior to use for coking. As fed to the jigs the distribution of the coal is :—

Over 1 in.	30 per cent.
1 to ¼ in.	45 ,,
¼ in. to 0	25 ,,

The coal of all three sizes is mixed after washing.

The average results obtained during the year 1923 were as follows :—

	Ash per cent.	Sulphur per cent.	Moisture per cent.
Feed coal	11·4	2·75	6·0
Washed coal	7·4	1·9	13·6
Refuse	38·4	8·4	—

The efficiency of the removal of dirt from the coal may be judged by the results of float-and-sink tests, using a liquid of S.G. 1·35, as follows :—

	Weight per cent.	Ash per cent.	Sulphur per cent.
Sinkings in washed coal	4·8	27·2	4·2
Floatings in refuse	33·5	7·9	2·0

These results show that a third of the material discharged as refuse consists of good quality coal containing only 0·5 per cent. more ash and 0·1 per cent. more sulphur than the washed coal. At the same time nearly 5 per cent. of the washed coal consists of material containing 27 per cent. of ash and 4·2 per cent. of sulphur. And this is in washing sized coal. It is safe to say that many questions would be asked if similar results were shown by a modern Baum washer in England which, moreover, would have to deal with unsized coal.

Incidentally, these results offer a good example of the utility of float-and-sink tests. The reduction of the ash content by one-third to 7·4 per cent and of the sulphur content by nearly one-third to 1·9 per cent. might be passed, were it not for the state of affairs revealed by float-and-sink tests.

JIGS WITH PLUNGERS BELOW THE SIEVE

One type of jig, like the Meguin washer previously described, has the plunger situated below the sieve. The simplest form of this appliance is that described by Rittinger and called by him the "Setzpumpe." Though it was never used to any extent for washing coal or concentrating ores, its simplicity is appealing. It is shown diagrammatically in Fig. 58.

Rittinger's Setzpumpe.—As the plunger, P, rises, the valves, V, in the plunger remain closed. The valves, S, in the false bottom open and allow a current of water to pass upwards through the

material on the sieve. The water falls around the rim of the sieve into the outer chamber carrying the lightest and finest material over with it. The upward current results in a stratification of the material on the sieve, which is rendered more complete during the downstroke of the plunger. During the downstroke, the valves, V, in the plunger open and allow the plunger to fall without disturbing the water above it. During this period the particles which were lifted from the floor of the sieve during the upstroke fall in relatively still water and tend to stratify according to density differences.

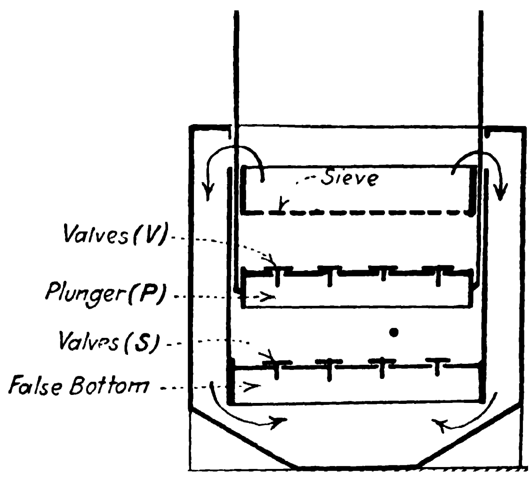

FIG. 58.—Rittinger's Setzpumpe.

It is with the object of producing or intensifying the effect of this phase of the movement of the particles that all jigs with plungers below the sieve and valves in the plunger head have been designed.

The Rittinger Setzpumpe was not successful in practical use on account of the troubles experienced in the valves and because it was intermittent in action. It was efficient, but was more troublesome than other equally efficient machines.

The Richards Jig.—Richards has also designed a jig to operate without any suction stroke. A constant head of water is admitted to the hutch of the jig below the sieve through a pulsator valve, so that the upward current of water through the sieve is applied

intermittently. Stratification occurs during the upstroke and is augmented as the particles subside in relatively still water whilst the pulsator valve is closed. The chief claim for this jig is that it economises space and water.

The Montgomery Jig.—The Montgomery jig, which has been used for coal washing in the United States, is fitted with a plunger working in a vertical sleeve under the jigging sieve, the plunger being operated by connecting rods passing through sleeves at the corners of the sieve and actuated by a lever and crank mechanism. In the surface of the plunger, flap valves are situated which open on the downstroke, but remain closed and cause an upward current through the bed of material on the sieve during the upstroke. Jigs of this type economise space, and, as a rule, require rather less power than jigs with a separate plunger compartment.

It will be noticed that many jigs used in America employ systems of automatic valves, which are situated either in the face or at the edges of the plunger, in the water-supply pipes, or over the water inlet below the jigging screen. Valves in the plunger are not always certain of action, and trouble is frequently experienced with them. All valves contained in the tank or hutch of the jig require frequent attention owing to corrosion, and, occasionally, clogging, due to the trapping of small hard particles of shale or pyrites. Rubber valves, as used in the Elmore and New Century jigs, are liable to deterioration and increase the friction.

Other Jigs.—In addition to those described, there are a number of other jig washers, the American, the Bilharz, the Baure, the Collom, the Conkling, the Diescher, the Evard, the Faust, the Flanchon, the Forey, the Forrester, the Gervais, the Girard, the Hancock, the Hodge, the Kasselowsky, the Lacratelle, the Marsais, the Meynier, the Neuerburg, the New Century, the Raxt-Madoux, the Revolier, the Rexroth, the Rivière, the Robert, the Schranz, the Shannon, the Skoda, the Stein, the Stutz, the Woodbury and many others. Few of these are of interest; many of them are no longer in general use, and the use of some of them, for example, the Bilharz, the Collom, the Conkling, and the Schranz, has been confined to ore-dressing practice. All the others are known, at some time or other, to have been used for washing coal. In addition to these there are a number of jigs which have been used exclusively for washing anthracite, among which the Christ, the James, the Lehigh Valley, the Reading and Wilmot-Simplex are the best known. There are, or have been, several others, designed for special or local requirements, but which have never come into general use.

CHAPTER X

UPWARD-CURRENT WASHERS: GENERAL

In describing the general theory of coal-cleaning in jigs and upward-current washers, (Chapter III) it was shown that a particle sinks in an upward current of water if the speed of the current is less than the terminal velocity of fall of the particle in still water, and rises if the current speed is greater than the terminal velocity of fall. Since the velocity with which a particle ultimately falls in still water is a function both of its size and specific gravity, there is a compensation between the size and density of particles of coal and of dirt, as a result of which large coal particles and smaller dirt particles of greater density than coal tend to move together. For efficient separation of the two types of material by an upward current of water, a preliminary grading according to size is therefore necessary.

An upward-current washer avoids the harmful effects of the downward current in a jig, and to that extent has an advantage over it. But it has the disadvantage that the upward current is continuously applied and is maintained at a constant speed. Consequently the advantages of pulsation in the jig are absent and, above all, the brief interval of time never arises, during which, because of the change of direction of the water current in a jig, the water and the particles are relatively still and the conditions arise for the separation of coal from dirt according to density differences and independently of size.

In competition with modern jig washers, the necessary preliminary sizing is the principal drawback of upward-current washers. This requirement tends to make the operation of the process more complicated and cumbersome, or, if previous sizing is not practised, less efficient. Consequently, for general purposes, a jig is usually preferred unless there is some specific advantage to be gained by the use of an upward-current washer.

One such advantage is the possibility of cleaning very fine coal more efficiently than is possible with a jig washer. With a coal rich in particles of an intermediate nature, consisting partly of coal and partly of shale, it is often necessary to crush the coal before it can be satisfactorily cleaned, and in these circumstances a large amount of fines may be produced. Draper, for example (*Proc.*, S. Wales Inst. of Eng., 1919, 35, 1), records figures for an Indian coal which required to be crushed to pass a ⅛-in. screen before it could be cleaned by any process. For this coal, the ability of the Draper washer to

deal with particles of very small sizes gave it an advantage over jig washers which are less able to treat very fine coal.

Upward-current classifiers have been used extensively in ore-dressing practice, but, in ore-dressing, the classification effected is regarded only as a preliminary stage in the concentration of the ore, to be followed by a finishing treatment. In coal-washing practice, on the other hand, upward-current classifiers are employed to yield the final washed product. This difference is not due to any difference in principle but to a difference in mechanical convenience, which centres, largely, in the greater throughput required of coal-washing equipment, and the fact that in ore-dressing practice, the value of the material treated justifies the use of a series of different appliances, each for a particular purpose. In coal washing, one plant is usually called upon to deal with the whole product, separate plants for separate sizes of material frequently involving too high a cost. In these circumstances, an upward-current washer, which is not expected to achieve the theoretical maximum efficiency, but only to remove the heaviest shale and clean the coal roughly without any preliminary treatment, can be installed and operated at a very low cost.

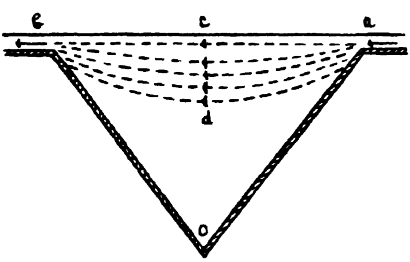

FIG. 59.—Diagram of Spitzkasten.

The earliest form in which the principle of classification by means of an upward current of water was employed was that of spitzkasten and spitzlutte. These appliances have been used for more than half a century for the treatment of metalliferous ores, chiefly for sizes which present difficulties in screening. They provide means of dividing minerals into classes of "equal-falling" particles which can then be submitted to suitable final dressing operations. Although, in ore-dressing practice, the size treated by spitzkasten and spitzlutte is that passing through $\frac{1}{25}$ in. (1 mm.), and exceptionally that below $\frac{1}{16}$ in. (1·5 mm.), spitzkasten were used in coal-cleaning practice for the classification of coal particles below $\frac{3}{8}$ in. prior to treatment in Lührig feldspar jig washers.

Spitzkasten.—The simplest form of spitzkasten is an inverted pyramid with an opening at its base. The principle upon which it acts may be seen by reference to Fig. 59. The material to be classified enters in a stream of water at *a*. The stream of water entering the box spreads out as shown in the diagram, and all the particles in the stream that are capable of falling through the belt of moving water at its maximum depth (*d c*) during the time taken to flow from *a* to *b*, descend to the bottom of the box and are discharged through the valve at the apex, *o*.

Spitzkasten were thus based entirely on free settling. As first used, the water within the boxes was stationary, and the particles therefore fell under conditions similar to those which Rittinger took as the basis of his theory. Within limits, the sizes and specific gravities of the particles settling in successive boxes were in accordance with theoretical calculations, but the setting up

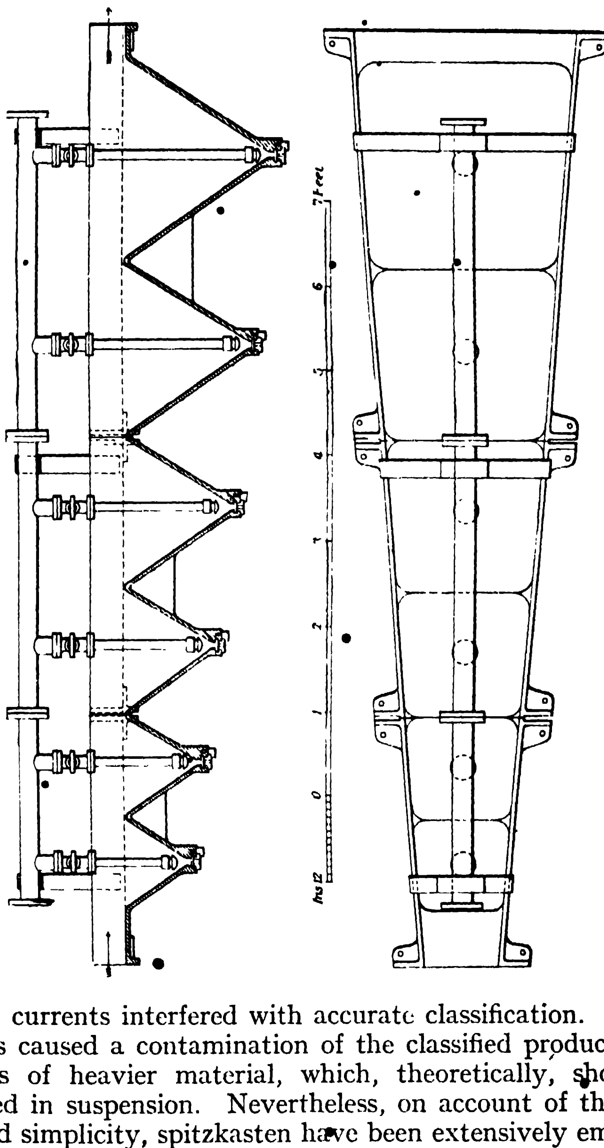

FIG. 60.—Cast Iron Classifier.

of eddy currents interfered with accurate classification. The eddy currents caused a contamination of the classified products by fine particles of heavier material, which, theoretically, should have remained in suspension. Nevertheless, on account of their cheapness and simplicity, spitzkasten have been extensively employed.

To overcome eddy currents, an upward water current was sometimes admitted, either through a vertical pipe extending from above

to the bottom of the box, or through a pipe inserted into the bottom of the box. Spitzkasten are usually arranged, as shown in Fig. 60, in the form of a continuously widening trough.

Spitzlutte.—A spitzlutte consists essentially of a V-shaped channel, the water with mineral in suspension passing down one limb (*a*, Fig. 61) and up the other limb (*b*). As usually constructed, it consists of two boxes, one in the shape of an inverted triangular prism within which a similar but smaller box is fixed. The position of the inner box is variable, so that the space between the two can be adjusted. In practice a series of connected V-channels are employed, the discharge at *b* passing into a second box. By using a second and third box with different widths of channel, the speeds of the currents in the later boxes differ from the speed in the first box, and so a fairly complete classification of the pulp supplied is effected.

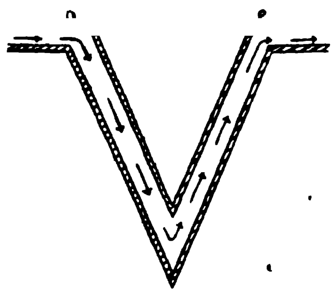

FIG. 61.—Diagram of Spitzlutte.

In the limb, *b*, the particles move in an upward current of water. Particles whose ultimate velocity of fall in still water exceeds the speed of the upward current fall to the bottom of the box, whence

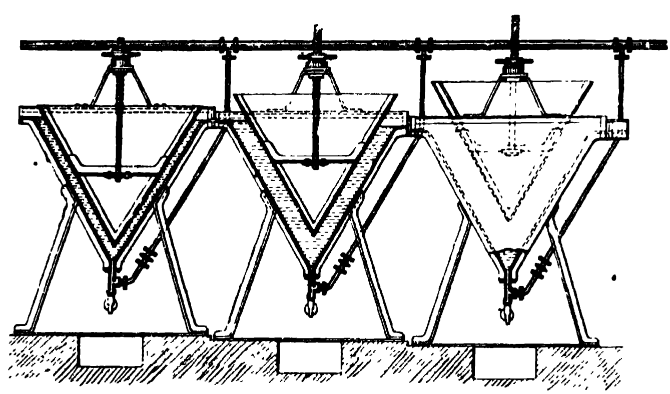

FIG. 62.—Cast Iron Spitzlutte.

they may be discharged. The classification effected by the spitzlutte is usually more completely in accordance with the ratio of "equal-falling" particles than is that effected by the spitzkasten, on account of the greater effective depth of water and a relative absence of eddy currents. An additional advantage is the possibility of adjustment, in that the width of the channels may be varied to suit particular circumstances. A form of spitzlutte is shown in Fig. 62 in sectional elevation.

"Hydraulic" water (as it is called) is admitted from the water main to the apex of the box through a valve or tap, and assists the classification. The amount of hydraulic water admitted is adjusted to control the speed of the upward current of water in the upcast limb.

Conical Classifiers.—A number of devices have been used in ore-dressing practice similar to spitzlutte, but composed of two concentric inverted cones. The inner cone tapered more steeply than the outer cone, and as the current of water passed upwards between the two, the narrowing channel between them prevented the speed of the current from falling as the cross-sectional area of each cone increased. The tendency for choking was therefore overcome, and owing to the relatively great depth of water, adequate opportunity was provided for the larger and heavier particles to be separated from the smaller and lighter. The usual method of feeding the particles was in a stream of water, admitted into the inner cone, whence the mixture passed from holes in the apex of the cone into the upward current of water admitted at the apex of the outer cone. The particles passing upwards with the water current were collected in a launder around the rim of the cone.

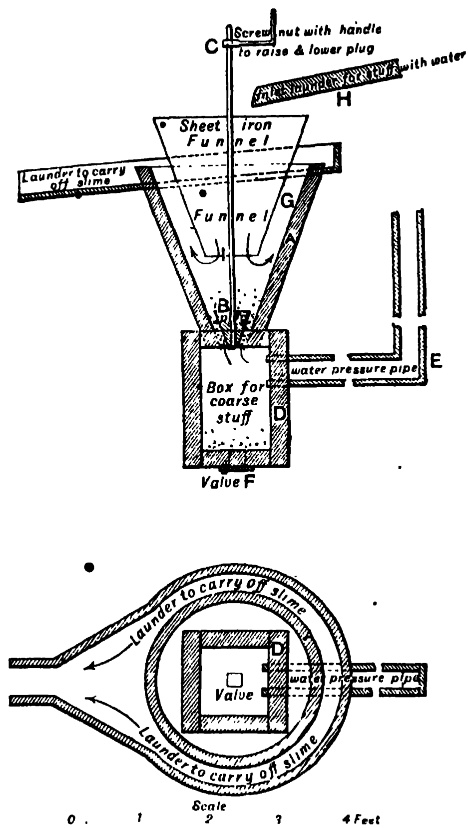

FIG. 63.—The Frongoch Classifier.

The Frongoch Classifier.—Classifiers of this type have been used extensively for ore-dressing in America and on the Continent. A primitive form of conical separator was used about forty years ago at the Frongoch mine in Cardiganshire, South Wales, for the treatment of blende and galena. It is shown in Fig. 63. The crushed ore was fed through the shoot, H, into the sheet-iron funnel, G, which was surrounded by a wooden cone, A. Water, supplied through E into the wooden box, D, caused an upward current, between the

cone and funnel, which carried upwards the small particles, but was insufficient to support the coarse, heavy particles. The coarse, heavy material fell into D, whence it was discharged continuously through the valve, F, consisting of a sliding iron plate. The width of the opening between the box, D, and the cone, A, was controlled by the plug, B, and the screw, C. Adjustment of the plug affected the speed of the upward current, and, therefore, the quality of the overflow. At the same time it controlled the discharge of coarse material into the box, D. At Frongoch, the coarse dis-

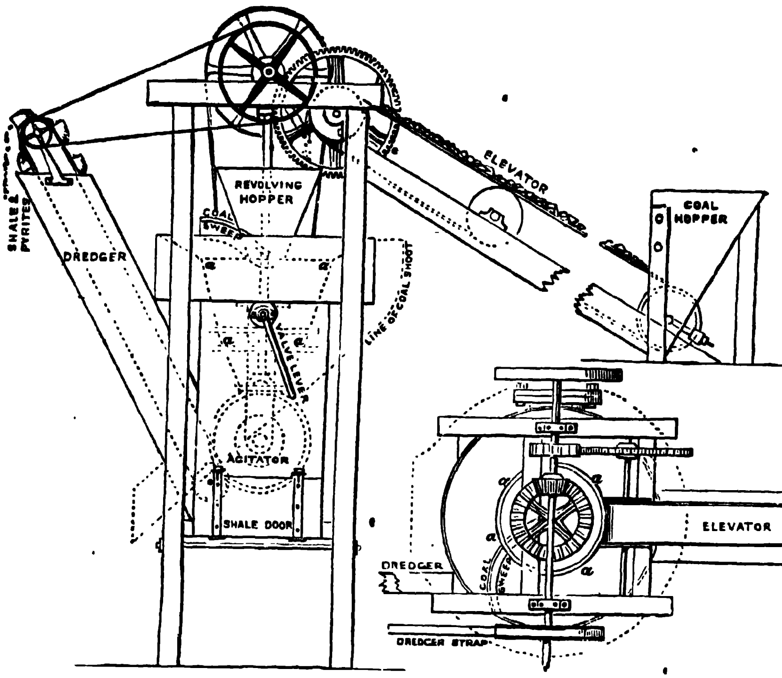

FIG. 64.—The Mackworth Coal Washer.

charge was further treated in jigs, and the overflow was passed to buddles for concentration.

The Mackworth Coal Washer.—The first upward-current appliance to be used for the washing of coal was the Mackworth washer invented in Great Britain in 1855. Groves and Thorp ("Chemical Technology," Vol. I., London, 1881) state that it was used in Scotland, Cumberland, Derbyshire, Gloucestershire and Wales.

The machine is illustrated in Fig. 64. The raw coal was fed into a revolving hopper, from which it fell by gravity into a conical vessel in which a current of water, supplied by the agitator at the lower end of the cone, ascended at a speed of about 4 to 5 ft. per

minute. The lighter coal, rising in the current, was swept by means of a curved rotating arm into a perforated shoot for delivery into the wagons. The dirt fell against the water current, through the valves, into a refuse-collecting chamber, from which it was removed by a dredger.

The valve through which the refuse fell consisted of an inverted metal cup partially closing the opening at the bottom of the cone. The amount of opening was regulated by raising or lowering the cup by means of a lever. The water draining from the washed products was returned to the agitator pump.

The Dor Coal Washer.—Another upward-current washer used for the washing of coal was the Dor washer installed at Ampsin (Henry, "Préparation mécanique des minerais de plomb," etc.; *Ann. des Mines*, 1871, 19, 294). The machine was used for washing small coal, but on account of its small capacity (0·4 ton per hour) and indifferent results it was considered to be unsuitable. It is illustrated in Fig. 65.

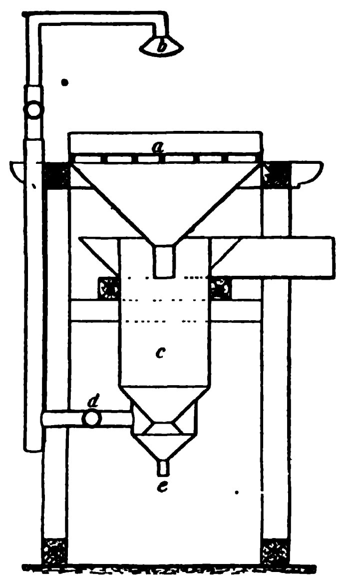

FIG. 65.—The Dor Coal Washer.

Better results were obtained at the Esperance Mine in Belgium, where the appliance was used for the treatment of sludge. The feed was placed on the sieve, *a*, and sprayed with water from a sprinkler, *b*. The material passing through the sieve fell through a funnel into the cylindrical chamber, *c*, where it was met by an upward current of water supplied through the valve, *d*. The coal was discharged at the upper periphery of the cylinder, the dirt falling through the conical bottom into the trap and valve, *e*, whence it was discharged. It is claimed that the ash content of the fine coal was reduced from 42·5 per cent. to between 12 and 15 per cent. The Dor washer did not, however, meet with much success, because of its low capacity, heavy water consumption, and, in sludge treatment, the high loss of coal.

The Stewart-Waldie Washer.—Although the Stewart-Waldie washer appears to have been erected at only one colliery in Great Britain, it introduced a novel idea, and was an improvement upon earlier types of upward-current coal-washer. It seems to have been efficient and to have had a high capacity (Waldie, *Trans. Min. Inst. Scot.*, 1887–88, 146).

The raw coal was fed through the iron guide, J, into a cylindrical

tank, A (Fig. 66), the floor of which was made of a perforated plate, with an aperture in the centre. The aperture was shielded by an iron hood, K, and the rotating arms, I. Through brass nozzles, H, fixed in holes in the perforated plate, a mixture of compressed air and water was pumped, and the upward current so produced raised the coal to the surface of the tank, whence it was removed through the overflow, M, to a revolving screen. The refuse was able to sink against the current, and fell downwards through the water, round

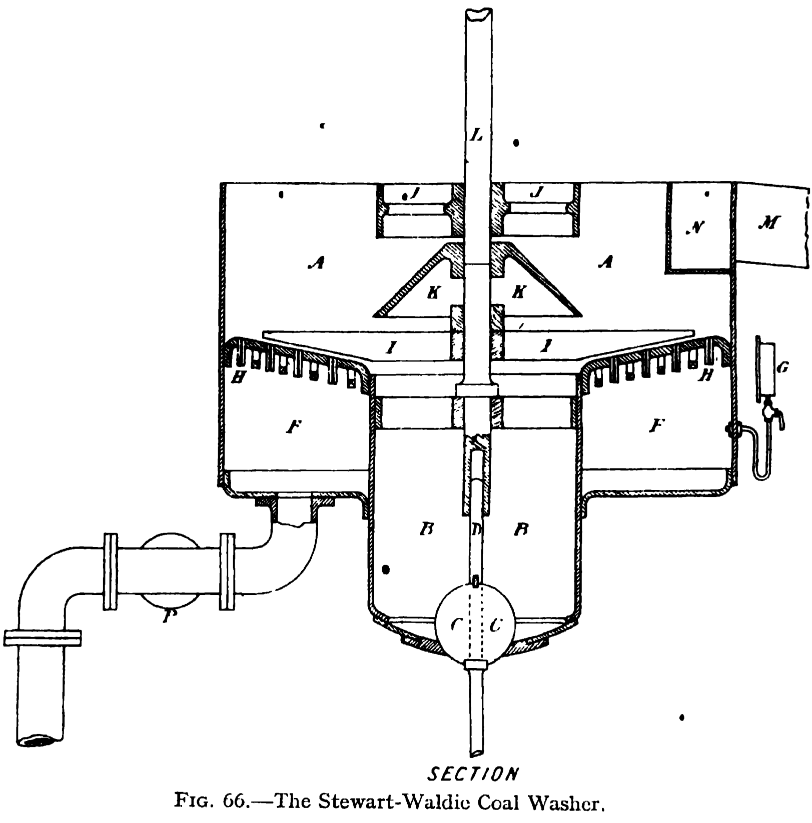

FIG. 66.—The Stewart-Waldie Coal Washer.

the hood, K, into the refuse chamber, B. The base of the refuse chamber was closed by a ball valve, C, which was guided by a sliding spindle, D, working in a space bored in the perpendicular shaft, L. The ball valve was opened and closed from outside by a lever.

The mixture of air and water was delivered from a double-acting pump, partially open to the air, into the compression chamber, F. The proportions of air and water giving the best results were about 1 to 1. The pump engine also rotated the shaft, L, and the rotating arms, I, attached thereto, at 20 r.p.m. The pump made forty strokes per minute and supplied the air-water mixture at 35 to 40 lb.

per sq. in. pressure, using 36 tons of water per hour, or $1\frac{3}{4}$ tons per ton of raw coal, the capacity of the washer being 20 tons per hour. It should be noted that the pump action, and therefore the production of an upward current, was intermittent, there being forty pulsations per minute. The pressure in the chamber, F, was recorded on the gauge, G, and was controlled by the safety valve, P.

The washed coal overflowed at M. The overflow was shielded by a plate, N, to prevent the discharge of pieces of shale carried round by the rotation of the stirring arms, I.

The results of washing were as follows :—

	Ash per cent.
Raw Coal	11
Washed Coal :	
Smalls	$4\frac{1}{2}$
Peas	4
Second Nuts	3
Third Nuts	$2\frac{1}{4}$

Generally the coal was sized before washing, but frequently unsized coal, through $\frac{1}{2}$ in. (gum) was washed.

The function of the air in the upward current was somewhat obscure. In the discussion on Waldie's paper it was stated that, if the air was omitted and water alone was used, the washing was spasmodic and unsatisfactory. When there was too much air, the surface of the water was violently agitated and, again, the washing was unsatisfactory. It was also pointed out that the use of the air reduced the effective specific gravity of the mixture providing the upward current, and it was correctly suggested that, on theoretical grounds, this would be a disadvantage. It was concluded that the effect of the air could not be otherwise than the provision of agitation in addition to that supplied by the revolving stirrers, and that this agitation must have been in some way beneficial.

A more probable explanation takes into account the intermittent working of the pump. It is possible that the compression chamber, F, acted as a reservoir, so that the upward current was not eliminated between pulsations, but merely slowed-up. In these circumstances the machine can be looked upon as similar in some respects to a jig, a strong upward current being followed by a slower upward current, instead of, as in a jig, a strong upward current being followed by a slow downward current. It may be possible by such an arrangement to effect a separation of two particles such that they cannot be separated in any one upward current of uniform velocity, but could be separated in a jig.

The fact that it was found in practice with the Stewart-Waldie washer that, to remove the supply of air (and so use an upward current of water only), resulted in unsatisfactory washing, does not rule out the possibility, for there is no evidence to show that the speeds of the various working parts were adjusted to suit the new

conditions. Moreover, in the absence of the air, the reservoir would not fulfil the same function. The water was admitted eccentrically and the speed of the upward water current would not be uniform over the area of the sieve; when compressed air was also admitted, it would help to distribute the water current more evenly.

If the pulsatory effect were the real reason for the apparent efficiency of the Stewart-Waldie washer, it would appear that beneficial results might be obtained in other upward current washers by some method of producing periodicity in the current more satisfactory than that used in the Stewart-Waldie. The principle of the joint admission of air and water has, however, been abandoned,

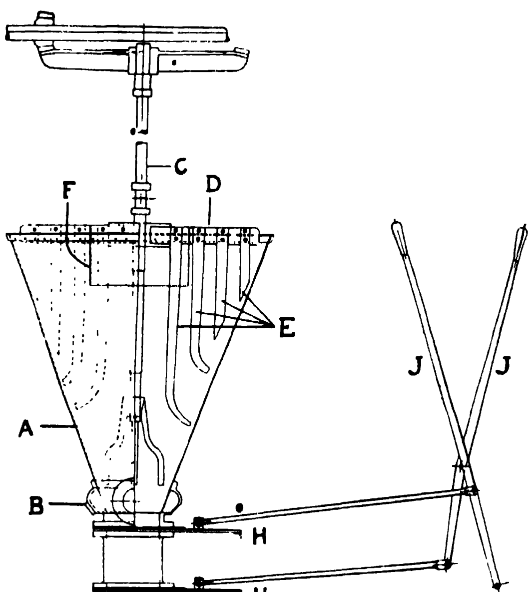

FIG. 67.– The Robinson Washer.

not only in Great Britain, but in America, where it has also been applied experimentally.

The Robinson Coal Washer.—The Robinson washer met with considerably more success than the earlier forms of upward-current washer. Its convenience for the washing of coal is evidenced by the fact that Robinson washers are still being erected. It was first introduced about 1885 at some of the collieries of Messrs. Bolckow Vaughan, of which firm Robert Robinson was the mining engineer.

The washer is illustrated in Fig. 67. The left half of the diagram shows an elevation of the washer, the right half a section through it. The appliance consists of the inverted frustum of a cone, A, constructed of cast iron or steel $\frac{3}{8}$ in. thick. One of the earliest washers, which had a capacity of 20 tons per hour, was 8 ft. in diameter at

the upper end, 1 ft. 10 in. in diameter at the lower end, and 6 ft. 6 in. deep.* A modern plant, to deal with 30 to 40 tons per hour, employs a cone 10 to 12 ft. in diameter at its upper end, 1 ft. 10 in. at the lower end, and 8 to 10 ft. deep. The lower end of the cone is surrounded by an annular chamber, B, perforated on the inside with several rows of ½-in. holes, so that water can be forced up through the cone in a large number of jets. Water is fed to the annular chamber under pressure, and the water overflowing at the top of the cone is collected and pumped back to the supply tank.

A vertical driving shaft in the axis of the cone carries four oak cross-beams, D. From each of these wooden arms, wrought iron bars, E, project downwards into the body of the cone so that they almost touch its sides. At the bottom of the driving shaft, four shorter bent arms are carried. In operation, the shaft is rotated by a crown wheel and pinion drive at a speed of 10 to 14 r.p.m., causing the water to acquire a circular motion in addition to its upward movement. The mass of particles in suspension in the water during the washing process is therefore agitated. The coal is fed into the cone from a shoot, or directly from an elevator, into the centre of the cone and inside an iron baffle ring, F. The baffle ring is attached to the cross beams, D, and its lower end is immersed about 15 in. below the level of the water in the cone. To reach the discharge point the coal must pass under the iron ring, and in so doing, each particle comes properly under the influence of the current of water in the cone, and small particles or agglomerations of particles are prevented from floating along the surface and being discharged. In some plants a second baffle ring, projecting about 12 in. below the water surface, is attached near the ends of the cross beams.

The raw coal is separated by the upward current into washed coal and refuse. The washed coal overflows at the discharge point, a lip cut in the rim of the cone, and the refuse sinks through the water into the refuse chamber. The refuse is discharged by sliding plate valves, H, H, operated by the levers, J, J. The upper plate is opened to allow refuse to collect in the chamber. It is closed periodically and the lower plate removed to allow the refuse to fall into wagons or a dirt-removal elevator. This form of refuse-discharge valve is not altogether satisfactory, and in one South Yorkshire washery, with two cones, it is proposed to substitute a star valve for the continuous removal of the refuse from each cone into a screw conveyor.

In the older plants, the water supplied to produce the upward current in the washing pan was stored in a tank 30 ft. above the level of the water in the washer, and was supplied under a constant head of pressure. The water overflowing with the washed coal was collected by passing the washed product over a drainage screen, the water passing through the screen and into a reservoir, from which it was returned to the overhead storage tank by a pulsometer pump.

* Report of Coal-Cleaning Committee, *Trans. Min. Inst. Scot.*, 1889-90, 11, 145.

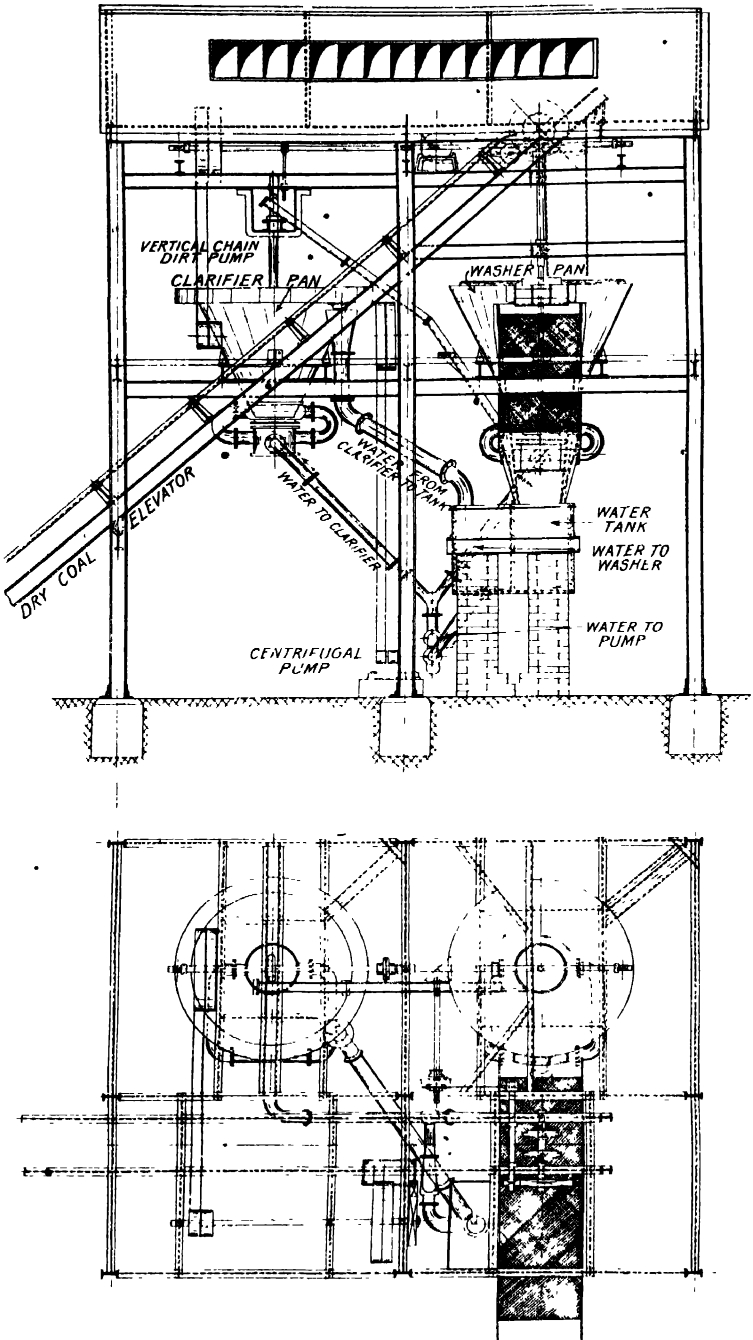

FIG. 68.—Arrangement of Robinson Washery. Front elevation and plan.

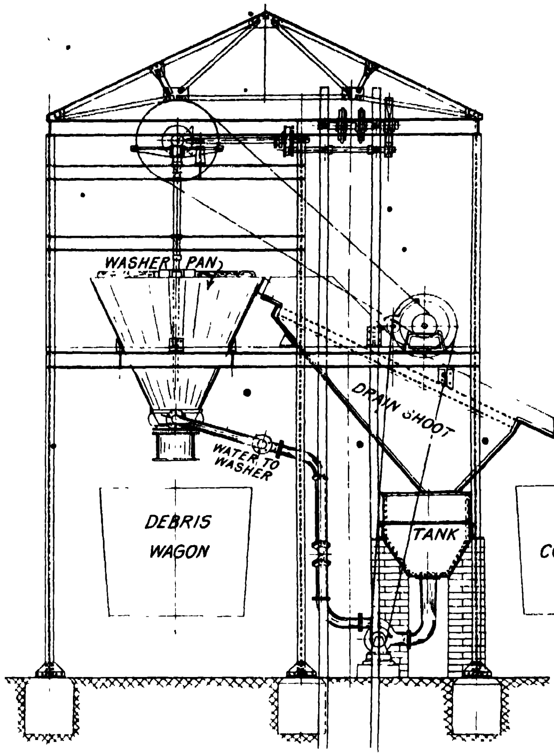

FIG. 68.—Arrangement of Robinson Washery. Side elevation.

The inconvenience of providing 30 ft. of headroom has led to the use of a circulating pump to supply the water under pressure to the base of the washer.

In all modern Robinson washeries a water clarifier is fitted, in which the particles suspended in the washing water are removed. This is desirable, not merely for the sake of efficient washing but also because of the wear and tear of the valves and pumps. The amount of circulating water in a Robinson washer is high, and unclarified water materially increases the cost of upkeep of working parts. It is arranged that dirt particles with a minimum number of coal particles are deposited in the clarifier, and coal particles with relatively little dirt are deposited in the water storage tank. The clarifier is therefore made to act as a slurry refiner.

In British practice the clarifier (known as the Hargreaves clarifier) consists of a reservoir similar in shape to the washing cone. Indeed, in some cases an old 20-ton per hour washing cone is used as the clarifier for a 40-ton per hour washery. The clarifier is shown on the left-hand side of the front elevation in Fig. 68, which illustrates the lay-out of a typical one cone (25 to 40 tons per hour),

Robinson washing plant. Circulating water from the storage tank is pumped into a main leading to the washer and to the clarifier. The bulk of the water passes into the annular chamber at the base of the washer, but some of it is by-passed and enters the base of the clarifier. It slowly ascends in the clarifier, allowing many of the fine particles in suspension in it to settle to the bottom. The clarified water overflows slowly into a trough around the rim and is returned thence to the storage tank.

The water reaching the clarifier has already been through the washer and has been collected in the tank from the drainage screen. The particles in suspension in the water when it reaches the clarifier are those dirt particles which were too small to fall against the upward current of water in the washer, together with a number of fine coal particles. Because of the slower upward movement of the water in the clarifier the bulk of the dirt particles are able to fall out of suspension. The size of the particles entering the clarifier is, however, limited by the size of the holes in the drainage screen, and, whereas the dirt particles can settle out in the clarifier because of their high specific gravity, the coal particles, because of their lower specific gravity, are still floated by the slow upward current and are returned to the storage tank. Further settling of the fine coal particles takes place in the storage tank, where the water is relatively quiescent. The fine dirt settling at the bottom of the clarifying pan is removed by a vertical chain dirt elevator extending to the bottom of the cone along its axis. The dirt is delivered by a shoot to the wagons or to a dirt elevator at the base of the washing pan, and is removed, either with the large refuse from the washer, or separately.

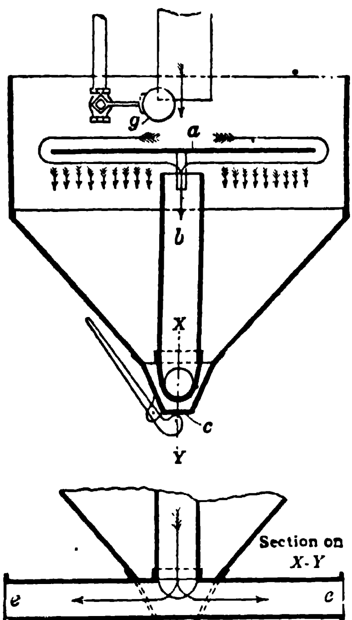

FIG. 69.—The Ramsey Sludge Tank.

A number of Robinson washers are in operation in America. The standard Robinson design has been modified in several particulars by another Englishman, Erskine Ramsey, who has fitted a more complicated form of clarifier, the Ramsey sludge tank, Fig. 69. The water containing suspended coal and dirt particles falls into a cone-shaped vessel near to the upper and wider end of which a horizontal plate, a, is suspended. The water is deflected round the edges of the plate and is removed by a vertical pipe, b, the upper

end of which is situated a few inches below the centre of the underside of the plate. The path of the water is indicated by large arrows. The suspended dirt particles fall out of the current of water, as shown by the smaller arrows, and are discharged from the base through the valve, *c*. The coal particles are carried forward in the water and travel into the vertical pipe, *b*, from which they are removed by connections at *e*, *e*, shown in the section on X–Y. The best separation of dirt particles is found by varying the diameter of the deflecting plate and the position of the vertical pipe. With some particular setting and dimensions the results are found to be better than with others, and for each particular case this setting, once found, is maintained. With too small a deflecting plate the impurities go forward with the coal particles; with too large a deflecting plate the coal particles fall out of suspension and are removed with the dirt particles.

The make-up water for the washing cone is admitted into circulation in the clarifying tank through a pipe controlled by the float valve, *g*. The water entering through *a*, together with this make-up water, is removed through the pipe, *b*, and is pumped back into the water circuit. In some American plants an additional innovation is a balancing column consisting of a standpipe, 80 ft. high and open at the top, placed between the base of the washing cone and the pump which drives the washing water into it. If a stoppage in the cone takes place, the pump forces the water up the standpipe and the increased pressure thereby provided is usually sufficient to break through the blockage and reproduce normal working conditions.

In practice it is necessary to maintain a regular rate of feed of coal to the cone of a Robinson washer. If the feed is too rapid, the cone tends to become choked and dirt particles will pass over with the coal, whereas, with too slow a feed, the proper accumulation of refuse particles forming a kind of bed at the bottom of the washer is lost. The regularity of feed is assured by conveying the coal in bucket elevators to the feed shoot, whence it falls into the centre of the washer.

Reverting to Fig. 68, which shows the lay-out of a single cone washery in front and side elevation, and in plan, it will be seen that the operation of the process is quite simple. The raw coal is elevated and delivered by a shoot (not shown in the diagrams) into the washer, whence the cleaned product and the refuse are discharged directly into wagons. It will be noticed that no provision is made for screening the coal either before or after washing. The Robinson washer is used for unsized coal, but the results are more satisfactory if the coal is screened before washing into the sizes required in the product and each size is then put through the washery separately. Although all coal from 2 in. may be washed at one and the same time in a Robinson washer, if the products are to be sold as nuts, beans and slack, the best results will be obtained if the nuts, the beans and the slack are each washed separately. The flow of the water is shown

clearly in the front elevation, and the drainage screen is shown in the side elevation.

A washery with a throughput of 40 tons per hour is usually driven by two motors, each of 25 h.p. One motor drives the pump, circulating about 30,000 gallons of water per hour. The other drives the elevator, the revolving stirrer in the washing cone, and the chain dirt elevator.

When the Robinson washer was first introduced, it was found to be unsatisfactory for unsized coal, and, as a rule, the coal was crushed and only small coal, below about ¾ in., was washed. The washing was often effected without any further grading of the coal below ¾ in. size, but the usual practice was to screen the small coal into two sizes. The Mining Institute of Scotland Coal-Cleaning Committee, who reported their findings in 1889–90, described several Robinson washeries in operation in Scotland, Yorkshire and Lancashire. Their records were rather incomplete, but it appears from their findings that the necessity of sizing was amply recognised. They state (*loc. cit.*, p. 162) that : " The coal being reduced to a uniform size previous to washing, the Robinson machine gives satisfactory results." In none of their descriptions, however, do they state the sizes washed. In one case they report (*loc. cit.*, p. 203) that a bash washer (known as " Bell's ") allowed a considerable loss of fine coal in the dirt and recommended that the " dross " be sized before washing, and the small sizes be washed " in a more suitable machine, such as Robinson's, or perhaps, better, a bash washer, with the use of feldspar on the top of the plates." Robinson's washer appears, therefore, to have been considered more suitable for washing small coal than an ordinary piston jig washer without a false bed.

In more recent times it has become the practice to wash coal of larger size than ¾ in. in a Robinson plant, 2 in., or even 2½ in. coal being washed unsized. The only modification introduced to enable this to be done efficiently is the adoption of the clarifier for removing the fine dirt from the washing water. Nevertheless, although this is the practice in certain cases, it is safe to say that it is only possible in a limited number of instances, in which, owing to the distribution of coal according to size and the distribution of the mineral matter in the sized particles, washing is comparatively easy. With a coal containing a considerable bulk of particles of density intermediate between that of coal and shale, or one with a high concentration of mineral matter in the smaller sizes, the Robinson washer would be unsuitable without a preliminary grading of the raw coal, or unless only coal below, say, ¼ in. were washed.

In one plant, where coal below 2½ in. is washed in one operation, it is stated that the washed products are of a satisfactory quality, though complaints are not infrequently made with regard to the quality of the smudge (the washed coal through ⅝ in.). To ensure satisfactory cleaning of the small coal it is necessary to wash it

separately. This is in accordance with the conclusions to be drawn from a consideration of the theoretical size limits for separation, which for coal of specific gravity 1·3 and shale of specific gravity 2·5 are 5 to 1. Coal of 2½ in. size would therefore be contaminated with all the dirt below ½ in. Actually the ratio of 5 to 1 may be rather exceeded in practice, with a coal fairly free from intergrown particles, because the effective density of the washing medium is greater than unity, owing to the suspension in it of fine particles, and because the phenomenon of hindered settling, which increases the theoretical sizing ratio is, to a certain extent, brought into play. Even so, the smaller sizes of dirt still pass away with the washed coal, some of it, say that below $\frac{1}{30}$ in., being removed on the spraying screens.

The requirement for sizing a coal which is not easy to clean is the chief disadvantage of a Robinson washer, for though it has the advantage of a low installation cost and low working costs, and is compact and easy to operate, the compactness disappears if a number of cones must be erected, each to wash one particular size. Taking a capacity of 40 tons per hour per cone, there would be no disadvantage for a total capacity of 160 tons per hour in requiring four cones each to wash one given size, for four cones would be necessary to wash the unsized coal. But if a capacity of 40 tons per hour only is required, it would be a serious drawback if four cones, each occupying about 200 sq. ft. of floor space, were to be needed. Even if smaller cones were used, the floor space required would be reduced, but only by a very small amount. The other disadvantages of the Robinson washer are the high water requirements and the loss of small coal. For a single cone washery of 40 tons per hour capacity about 30,000 gallons of water are circulated per hour, about 3,000 of which passes through the clarifier and back to the water tank.

The capacity of the washer, and also the quantity of water required, depend upon the size of coal which is being washed. A cone with an output of 40 tons per hour of coal washed in one size from 2½ in. to 0, has only a capacity of about 30 tons per hour when washing slack below ⅝ in., and the amount of water required is proportionately reduced since the speed of the upward current of water is less for small coal than for large coal. The power consumption is about 1 h.p. per ton of coal.

The following results relate to the cleaning of coal by a Robinson washer at the Whitwood Colliery of Messrs. Henry Briggs, Son and Co., Ltd., and are published by permission of Major D. H. Currer Briggs.

	Liquid S.G. 1·4. Float.	Sink.
Raw coal	78·80	21·20
Washed coal	97·55	2·45
Refuse	5·86	94·14

clearly in the front elevation, and the drainage screen is shown in the side elevation.

A washery with a throughput of 40 tons per hour is usually driven by two motors, each of 25 h.p. One motor drives the pump, circulating about 30,000 gallons of water per hour. The other drives the elevator, the revolving stirrer in the washing cone, and the chain dirt elevator.

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	Liquid S.G. 1·4. Float.	Sink.
Raw coal	78·80	21·20
Washed coal	97·55	2·45
Refuse	5·86	94·14

The results are obtained in washing coal from 2½ in. downwards, and show that the washed coal contains 2·45 per cent. of removable dirt, and the refuse 5·86 per cent. of coal. This loss of coal in the refuse corresponds to a loss of 1·46 per cent. of the total feed coal.

In washing coal from 2½ in. to 0, the raw coal has an average ash content of 13·0 per cent., which is reduced by washing to 6·0 per cent. Slack below ½ in. before washing contains 21·0 per cent. of ash, which is reduced by washing to 6·5 per cent.

At another colliery, where coal below 2 in. is washed, the raw coal has an average ash content of 15 per cent., which in the washed coal is reduced to 5·5 per cent., with a loss of coal in the refuse amounting to 2 to 2·5 per cent. (reckoned as a percentage of the refuse).

The efficiency of the clarifier is illustrated by the following figures, which also relate to the Robinson washer with Hargreaves clarifier at Whitwood Colliery, where, of 30,000 gallons per hour of circulating water, about 3,000 pass through the clarifier.

	Composition of Solids.	
	Coal per cent.	Dirt per cent.
In water entering clarifier . . .	35·2	64·8
,, ,, leaving ,, . . .	60·0	40·0
In fine dirt removed from clarifier . .	6·9	93·1

Figures for a series of tests supplied by Messrs. Robert Wilson and Sons, Ltd., makers of the Robinson washer and Hargreaves clarifier, show that 1 cubic foot of water before clarifying contains 8½ lb. of solid matter, 3 lb. being removed by treatment. These figures are reckoned on undried solids. When dried, 1¼ lb. of fine solids is removed per cubic foot of water. On a 40-ton per hour plant this is equivalent to the removal of 625 lb. of fine dirt per hour. On the basis of dry solids the figures given correspond to a concentration of 5·58 per cent. of solids in the water before clarifying, and 3·62 per cent. after clarifying. A test at Whitwood Colliery showed that the clarified water contained 4·43 per cent. of suspended solids, whereas the water leaving the washer contained 6·85 per cent., corresponding to a deposition of 1·51 lb. per cubic foot of water between the washer and the clarifier exit. If 1¼ lb. is removed in the clarifier, ¼ lb. (one-fifth as much) is deposited on the bottom of the storage tank.

Further results for the cleaning of coal below ⅝ in. are given in Table 64. These results are the averages of two separate tests. The raw coal and washed coal were sized after washing into six sizes and tested in solutions of S.G. 1·4, 1·5 and 1·6.

TABLE 64.—RESULTS OF WASHING $\frac{5}{8}$ IN. TO 0 COAL IN ROBINSON WASHER. AVERAGE OF TWO TESTS

Raw Coal

Size. In.	Fraction. Per cent. of total.	S.G. < 1·4.	S.G. 1·4—1·5.	S.G. 1·5—1·6.	S.G. > 1·6.
Over $\frac{1}{4}$	21·5	82·0	1·5	0·9	15·6
$\frac{1}{4}$ to $\frac{1}{8}$	23·0	80·9	1·7	1·0	16·4
$\frac{1}{8}$,, $\frac{1}{16}$	25·0	72·3	1·6	1·2	22·9
$\frac{1}{16}$,, $\frac{1}{32}$	13·2	64·2	2·4	1·9	31·5
$\frac{1}{32}$,, $\frac{1}{64}$	8·6	63·7	6·9	1·9	33·4
Through $\frac{1}{64}$	8·7	49·7	6·2	10·6	33·5
Total	100·0	72·3	2·5	2·0	23·2

Washed Coal

Size. In.	Fraction. Per cent. of total.	S.G. < 1·4.	S.G. 1·4—1·5.	S.G. 1·5—1·6.	S.G. > 1·6.
Over $\frac{1}{4}$	17·9	98·8	0·4	0·7	0·1
$\frac{1}{4}$ to $\frac{1}{8}$	30·2	96·7	0·9	1·2	1·2
$\frac{1}{8}$,, $\frac{1}{16}$	31·7	93·8	1·3	1·5	3·4
$\frac{1}{16}$,, $\frac{1}{32}$	7·8	94·2	0·9	1·4	3·5
$\frac{1}{32}$,, $\frac{1}{64}$	6·4	81·4	5·6	2·8	10·2
Through $\frac{1}{64}$	6·0	51·7	3·9	22·9	21·5
Total	100·0	92·3	1·4	2·7	3·6

This comprehensive series of results show that the coal is very well cleaned down to about $\frac{1}{8}$ in., but below that size the cleaning is considerably poorer, and in the smallest size scarcely any improvement is effected.

The results for the raw coal are interesting, for they show clearly that the smaller sizes are much dirtier than the larger, there being a progressive increase in the amounts of sinks at S.G. 1·6.

CHAPTER XI

UPWARD-CURRENT WASHERS: THE DRAPER WASHER

The Draper Washer.—The most recent upward-current washer is that invented by the late Mr. J. M. Draper, and although as yet it has found only a small field of employment, its simplicity and flexibility suggest that, for certain purposes, its use may become more general.

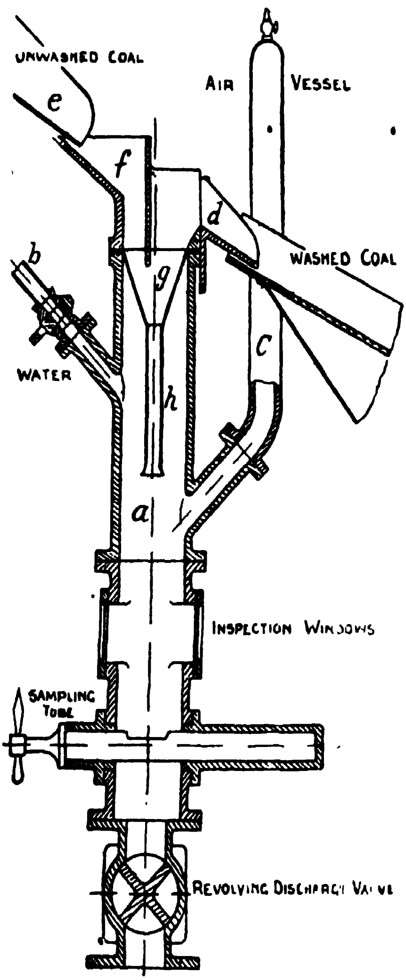

FIG. 70.—The Draper Washing Tube: Early Form.

The advantages which it has over many other types of washer are its compactness and its ability to wash small coal. In a Baum or other jig washer, which is designed to wash large quantities of coal, and frequently without previous sizing, the small coal is only cleaned indifferently. In a Draper washer, coal can be cleaned in sizes down to $\frac{1}{100}$ in., and this is a great advantage, for the smaller sizes of coal may be the dirtiest, and the disposal of dirty small coal is a matter of some difficulty.

The Draper washer in its earlier form was described by Knox (*Proc.*, S. Wales Inst. Eng., 1918, 34, 3, 291). It consisted of an inverted cone, *g* (Fig. 70), attached to a straight length of pipe, *h*, enclosed within a vertical steel column, *a*. The feed coal, which was sized before cleaning, was delivered into the hopper, *f*, from the shoot, *e*. As it fell into the cone it met an ascending current of water which separated the coal into two fractions, the lighter portion being discharged at the sill, *d*, the heavier portion falling into the pipe, *h*.

For the earlier form of washing tube, the device shown in Fig. 71 has been substituted. The principal changes effected are in the feeding and discharging arrangements, the direction in which the inlet water is admitted, and the dimensions of the tubular extension pipe. The coal is now fed from a shoot, E, into a central funnel-shaped hopper, F, from which the coal falls into the washing tube, H. On the outside of the neck of the feeding funnel is a truncated cone-shaped sleeve, J, the position of which is adjustable, the sleeve being clamped at any desired position on the neck by means of four

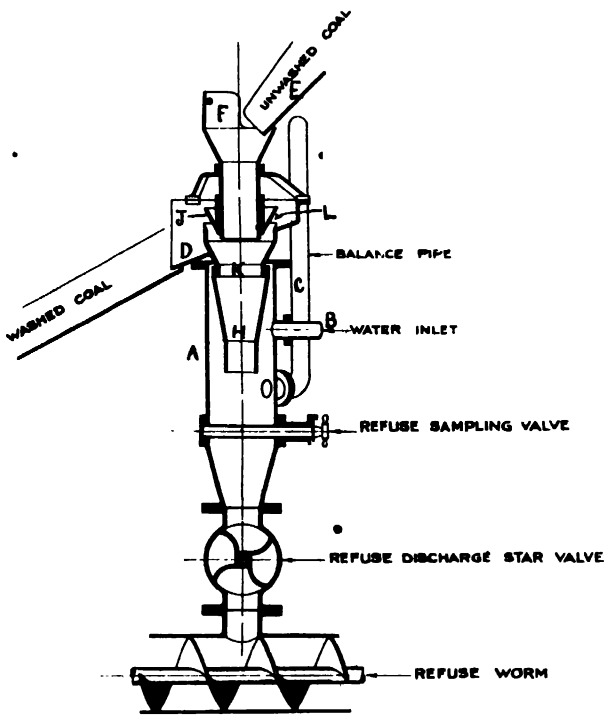

FIG. 71.—The Draper Washing Tube: Latest Form.

bolts. The funnel-shaped mouth, K, of the washing tube, H, is fixed in position, so that by sliding the movable cone, J, down the neck of the feeding funnel the width of the aperture, L, at which the washed coal is discharged into the trough, D, can be reduced. By sliding the movable cone upward, the width of the discharge aperture can be increased. The conical member, J, acts as a local restriction, its position being so determined that the velocity of the ascending current of water is increased at the discharge point. All coal particles which are carried upward in the current towards the discharge point are therefore carried over rapidly by reason of the locally enhanced speed of the current.

The raw coal falling from the hopper, F, into the tube, H, meets the ascending current due to the water admitted at B. In the earlier form, the usual width of the tube, *h* (Fig. 70), was 4 in. for coal from $1\frac{1}{4}$ in. down to $\frac{5}{8}$ in. in size, and 2 in. for coal below $\frac{5}{8}$ in., and the tube was straight sided. Experience with the later washing tube has suggested the use of 6 in. diameter tubes for $1\frac{1}{2}$ in. to $\frac{3}{8}$ in. coal, 5 in. tubes for $\frac{3}{8}$ in. to $\frac{1}{16}$ in. coal, and 4 in. tubes for coal below $\frac{1}{16}$ in., and the shape shown in Fig. 71. The fitting of wider tubes to the improved washer enables the same degree of cleaning to be effected whilst allowing a greater throughput and making it possible to wash larger sized nuts.

Instead of admitting the water in a direction somewhat inclined to the axis of the washer, as in the older form, the water is now admitted horizontally and tangentially, as shown in Fig. 72 (which shows the Draper washing tube in plan), so that the upward current of water produced in the washing tube has a spiral motion instead of a direct vertically upward motion. It was stated by Draper (*Proc. S. Wales Inst. Eng.*, 1919, 35, 1) that the gyratory motion of the water caused foliated shale particles to turn over, and fall more rapidly than they would otherwise do. As a result it was found possible to remove flat plates of calcite and calcium sulphate entirely from the coal even in the smallest sizes.

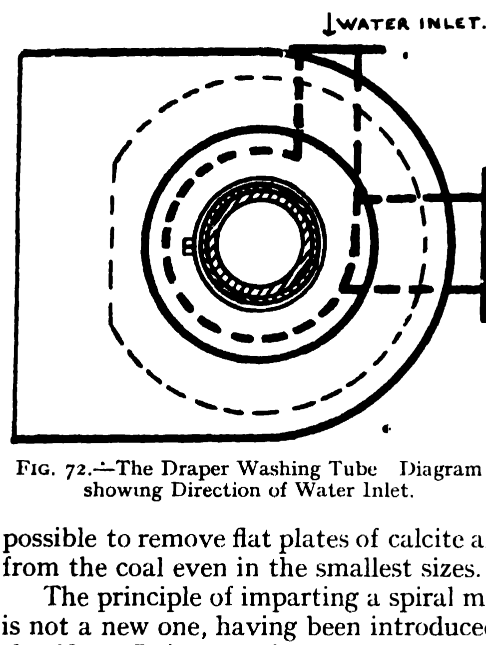

FIG. 72.—The Draper Washing Tube Diagram showing Direction of Water Inlet.

The principle of imparting a spiral motion to the upward current is not a new one, having been introduced by Richards in his Vortex classifier. It is operative to some extent in the cone classifier previously described, and also in the Robinson washer. In each of these appliances, however, it is incidental and slight. In the Robinson washer the slow rotation of the stirring arms produces no degree of swirling comparable with that in the modern Draper washing tube, and its chief function is to break up agglomerations of particles and to carry the cleaned coal round the washing cone to the discharge point. For cleaning coal, the production of a spiral movement in the ascending water current no doubt has the advantage suggested by Draper of causing foliated shale particles to fall, but its beneficial effect is probably to be attributed to a considerable extent to the fact that it eliminates the eddy and counter currents

set up in a turbulent flow of water straight up the washing tube. Eddy and counter currents tend to cause a contamination of the products, coal being trapped amongst the refuse and dirt amongst the coal, and their elimination makes for a greater efficiency.

In both the old and new forms of Draper washer, the water below the bottom of the washing tube (*h*, Fig. 70; H, Fig. 71) remains comparatively still, and the particles of refuse falling out of the washing tube descend into relatively quiescent water in which they fall rapidly to the bottom of the main tubulure. They are discharged by a water-sealed revolving star discharge into a screw conveyor, which collects the refuse from a battery of tubes and delivers it into the refuse hopper.

Above the refuse-discharge valve is a sampling device consisting of a sliding tube with a cup-shaped depression fitted into a suitable chamber. Some of the particles falling to the bottom of the apparatus are arrested in their fall and are collected in the cup, whence they can be removed by withdrawing and inverting the sampler. By this means the refuse can be inspected at any given moment and examined by float-and-sink methods to discover to what extent it contains material which it is desired to recover. The coal discharged at the circular rim of the funnel-shaped mouth of the tube, K, falls into the trough, D, where it is readily accessible for inspection and sampling for float-and-sink analysis. By these means the efficiency of washing is checked from time to time, and, when necessary, appropriate adjustment of the water supply is made. For convenience of operation under varying conditions, the valve regulating the water supply is fitted with a handle which moves in front of a graduated scale and enables the approximate setting for known conditions to be attained readily.

As a further approximate check on the efficiency of operation, glass inspection windows were formerly fitted so that the refuse falling in the lower part of the tube could be seen, but as the glass was soon abraded, they were of limited utility and have been omitted from the latest design.

A subsidiary tube, C, is connected to the main tubulure, A, below the water inlet and just below the bottom of the inner separating tube, H. This tube is closed at its upper end and serves as a balancing column in the event of the discharge cone or the separating tube being choked by an irregularity in the feed. If a stoppage takes place in the separating or discharging zone of the washer, there is a momentary check to the upward water current, and to overcome the choking an increased pressure of water is required. It would not be convenient to alter the setting of the water supply valve to overcome a temporary choking, so some automatic device was required for the purpose, and this was provided by the air column, C. When choking occurs, water is still supplied to the washer and causes an increased pressure in the air column. The increase in the applied pressure thus provided is, as a rule, sufficient

to free the stoppage, and restore the normal passage of the water current. When this is restored, the water level in C falls again to its normal height. In the older apparatus the air vessel (*c*, Fig. 70) had a tap at its upper end, but this has been omitted in the latest design.

Fig. 73 is a photograph taken inside a Draper washery and supplied by Messrs. The Pearson and Knowles Coal and Iron Co., Ltd. On the extreme left of the photograph the full height of the balancing column (C, Fig. 71) can be seen. A battery of seven tubes is shown in the centre of the photograph, and on the tube on the right and near end of the battery, the water inlet, the balancing column joint, and the refuse-sampling device can be seen. Below the sampler is the plate sealing off the inspection windows. At the top of this tube the inclined launder receiving the washed coal (D, Fig. 71) can be seen, and just showing above this is the top of the adjustable conical sleeve (J, Fig. 71).

Across the top of the photograph is the water main with the supply pipes branching downwards. The water regulating valves, with handles moving on a graduated scale, and the means of admitting the water from the supply pipes into the Draper tubulure in a horizontal and tangential direction, can also be seen. In the right foreground is the drive and shafting for the refuse star discharge valves, and the bevel gearing of the valves is shown on the battery on the extreme right of the photograph.

Until the early part of 1926 the Draper washer had only been in operation on a semi-industrial scale, the plants erected by the makers, Messrs. Sheppard and Sons, Ltd., in South Wales, and Messrs. The Pearson and Knowles Coal and Iron Co., Ltd., in Lancashire, having been employed for experimental purposes rather than for commercial production. In the spring of 1926, however, a plant was erected by Messrs. The Pearson and Knowles Coal and Iron Co., Ltd., at their Coppull Colliery, near Wigan, to wash 35 tons of coal per hour. The washing equipment consists of twenty-one tubes arranged in three batteries, each containing seven tubes. The lay-out is shown in Fig. 74.

The coal, which includes all sizes below 1½ in., is tipped from wagons into the underground receiving hopper, A, by hydraulic tippers, C, C, and is conveyed by a scraper conveyor, B, to the buckets of the feed elevator, D. The rate of supply by the conveyor to the elevator is controlled from the surface by sliding gates to ensure a regular rate of supply of coal to the washery. The elevator is totally enclosed for protection against the weather. The coal is discharged into the main revolving screen, E, which is of the single shell type, but is composed of three sizes of wire mesh screen to separate the coal into three fractions, namely, 1½ to ⅝ in. (double nuts), ⅝ to ⅜ in. (single nuts), and ⅜ in. to 0. The two larger fractions fall into hoppers, from which they are passed to the washing tubes by scraper conveyors, G. The fraction through ⅜ in. falls on to

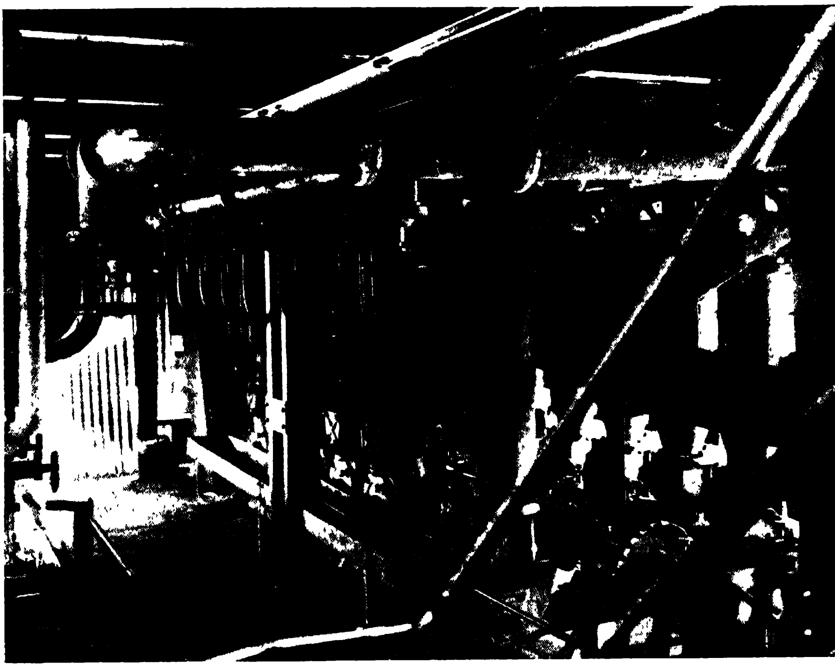

FIG. 73 Battery of Draper Washing Tubes.

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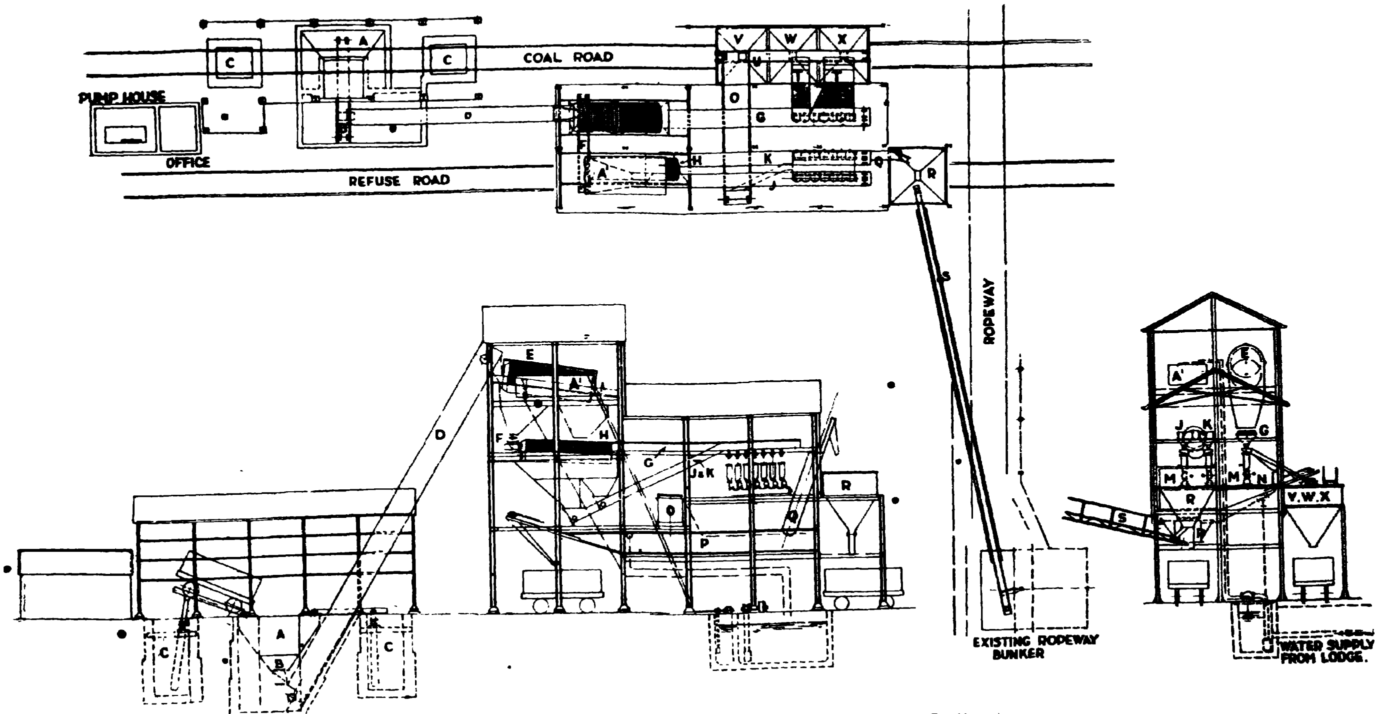

FIG. 74.—Arrangement of Draper Washery (Coppull Colliery).

The scheme of operation and water requirements of the washing plant were decided after a series of experimental trials. After determining (*a*) the distribution of the coal according to size by repeated sampling over a long period, and (*b*) the capacity of the washing tubes, it was possible to fix the number of tubes required for each fraction.

It should be remarked that the only difference between the washing tubes for the larger sizes of coal and the tubes for the smallest sizes is the diameter of the internal pipe (H, Fig. 72), the outer casing and other members of the apparatus being of uniform dimensions. For small coal a narrower tube is required than for large coal, and since the upward current of water is also slower, the capacity of the small coal washers is accordingly reduced. Thus, for 1½ to ⅝ in. coal, each washing tube can treat 3 tons of coal per hour, and the seven tubes washing the two largest sizes handle a total of 50 per cent. more coal per hour than the fourteen tubes washing the four smaller sizes. For coals other than that for which the washery was designed, the number of tubes required for the same output would vary according to the size distribution of the feed coal; more or less tubes would be required according as the coal contained a larger, or a smaller, proportion than 60 per cent. of coal through ⅜ in.

The capacity of each individual Draper washing tube is small, but the capacity of a battery of tubes per square foot of floor space occupied compares favourably with the requirements of many other washers; particularly is this the case when washing coal of greater size than ⅜ in. Thus Draper (*Proc.* S. Wales Inst. of Eng., 1919, 35, 1, 21), claims that a capacity of 5 to 6 tons per hour can be attained with a single washing tube with a 12 in. diameter base. (Presumably this claim applies to the cleaning of large nuts. As has been stated, the capacity on smaller sizes is considerably less.)

The over-all power requirements for the washery at Coppull Colliery, which is designed to deal with 35 tons per hour of raw coal of sizes below 1½ in., total about 70 h.p., though 122 h.p. has been provided, as follows:—

	h.p.
Main drive	60
Dirt creeper	6
Main pump, 1,600 gallons per minute, elevated 65 ft.	45
Fresh water pump, 80 gallons per minute, elevated 65 ft.	5
Hydraulic ram (tippers)	6
	122

To operate the whole washery, two men and one boy are a sufficient labour provision.

Table 66.—Washing Results, Experimental Draper Washer

	Ash Contents per cent.		
	Raw Coal.	Washed Coal.	Refuse.
Cap. Coll. Co. Slurry, $\frac{1}{8}$ in. to 0	23·7	7·2	70·6
M. Main Coll. Co. Do. $\frac{1}{25}$ in. to 0	13·3	4·2	67·4
Do. Belt pickings, crushed	35·8	8·6	71·7
B. Coll. Co., Ltd. Fines . .	31·0	5·7	71·2
M.H. Coll. Co. Do. . .	22·7	4·1	73·3
F. Coll. Co. Slurry . . .	30·2	4·8	74·0
Do. Do. . . .	18·4	5·2	70·1
Do. Do. . . .	16·0	5·0	62·9
Cl. Colls. Anthracite shale and rubbish	49·1	5·5	81·0
M. & Co. Waste slack . .	24·9	11·1	62·1
C.C. Co., Ltd. Fine coal . .	18·4	3·7	66·8
Ocean Coal. $\frac{1}{60}$ to $\frac{1}{100}$ in. . .	16·3	3·6	80·3
Do. $\frac{1}{10}$ to $\frac{1}{30}$ in. . .	15·4	2·4	80·0
Anthracite. $\frac{1}{10}$ to $\frac{1}{30}$ in. . .	11·5	2·0	88·5

Table 67.—Results of Washing in Draper Washer (Coppull Colliery)

Size (in.)	Per cent of Total	Ash Content per cent	Specific Gravity <1·4		1·4–1·5		1·5–1·6		>1·6	
			Wt per cent	Ash per cent	Wt per cent	Ash per cent	Wt per cent	Ash per cent	Wt per cent	Ash per cent.
Raw coal.										
$1\frac{1}{2}$–$\frac{5}{8}$.	24	17·3	73·0	4·4	6·5	17·6	6·0	27·0	14·5	77·4
$\frac{5}{8}$–$\frac{3}{8}$.	23	16·0	76·5	5·0	4·05	17·1	6·0	31·5	13·5	73·6
$\frac{3}{8}$–$\frac{9}{32}$.	12	19·5	73·0	4·6	4·0	18·1	4·0	29·6	19·0	74·8
$\frac{9}{32}$–$\frac{3}{16}$.	11	17·5	75·0	4·6	4·3	17·5	4·0	30·7	16·7	72·1
$\frac{3}{16}$–$\frac{3}{32}$.	15	17·5	77·5	5·3	2·3	19·2	2·6	24·7	17·6	70·6
$\frac{3}{32}$–0 .	15	27·4	65·0	4·8	4·5	20·0	2·8	23·9	27·7	75·0
Total	100	18·6	73·6	4·7	4·5	18·1	4·5	27·9	17·4	74·1
Washed coal										
$1\frac{1}{2}$–$\frac{5}{8}$.	36	6·8	89·4	5·0	7·4	16·8	2·2	28·0	1·0	49·6
$\frac{5}{8}$–$\frac{3}{8}$.	20	5·4	90·5	4·2	7·0	14·0	1·0	30·5	1·5	35·8
$\frac{3}{8}$–$\frac{9}{32}$.	15	6·6	90·7	4·0	4·0	15·8	2·7	26·0	2·6	62·2
$\frac{9}{32}$–$\frac{3}{16}$.	10	6·5	89·6	4·4	5·7	16·3	2·7	23·3	2·0	54·4
$\frac{3}{16}$–$\frac{3}{32}$.	9	8·2	86·5	4·2	5·6	17·3	2·9	24·7	5·0	57·8
$\frac{3}{32}$–0 .	10	16·2	74·0	4·4	6·7	17·6	3·6	26·6	15·7	69·0
Total	100	8·1	88·1	4·5	6·4	16·1	2·4	20·5	3·1	64·0
Refuse.										
$1\frac{1}{2}$–$\frac{5}{8}$.	18	66·6	0·3	7·8	*Nil.*	—	*Nil.*	—	99·7	66·8
$\frac{5}{8}$–$\frac{3}{8}$.	14	66·7	2·7	7·3	1·0	15·8	1·6	34·1	94·7	69·4
$\frac{3}{8}$–$\frac{9}{32}$.	9	65·2	*Nil.*	*Nil.*	2·3	10·8	2·7	23·3	95·0	67·7
$\frac{9}{32}$–$\frac{3}{16}$.	14	72·3	*Nil.*	*Nil.*	1·0	13·0	1·0	25·4	98·0	73·4
$\frac{3}{16}$–$\frac{3}{32}$.	18	73·1	*Nil.*	*Nil.*	0·5	11·0	1·2	27·3	98·3	74·0
$\frac{3}{32}$–0 .	27	71·9	7·0	11·0	2·3	9·7	2·0	30·9	88·7	79·0
Total	100	69·9	2·3	10·3	1·2	11·0	1·4	28·3	95·1	72·7

The possibility of washing slurry was examined by Draper, who obtained the results given in Table 66 in an experimental unit.

The first eleven results were given in an early paper by Draper (*Trans.* Ceramic Soc., 1917–18, 17, 213), and the last three in his later one (*loc. cit.*). The results do not show the amount of shale (sinks) in the washed coal nor the amount of coal (floats) in the refuse, but obviously, if there was any contamination at all, its amount could only have been slight. Especially with the last three, the separation effected was probably as nearly perfect as is possible with any process. It is worthy of notice, moreover, that in the first of these three examples, the coal all passed through a $\frac{1}{60}$ in. sieve, and, under these conditions, the cleaning results are excellent.

The results of washing at Coppull Colliery are given in Table 67. Float and sink tests were done on the raw coal, the washed coal and the refuse, each being first sized into six fractions.

The Richards Vortex Classifier.—In describing the Draper washer, reference was made to the Richards vortex classifier in which the upward water current is made, by design, to travel with a spiral motion. The classifier consists of a trough with a series of vertical separating columns in its base, and is used for the dressing of mineral ores A current of water rising through the columns in a helical path causes the material to be separated into two fractions, the one rising and passing into the next column, and the other falling. The spiral motion is acquired by the admission of water through vortex fittings into a hydraulic chamber at the base of the column, and its effect is to remove any tendency to produce a strong upward current in one portion of the column, carrying particles of the heavy material with it, whilst a downward or weak upward current is produced in some other portion, allowing light particles to contaminate the discharge products.

With the spiral motion, there is a possible tendency for light grains to descend in the comparatively idle water in the axis of the stream. If this is encountered, a central core is suspended in the axis of the column, so ensuring a more complete spiral.

The horizontal component of the helical upward current is not effective in causing the light particles to rise to a higher position than the heavy particles, and it is doubtful if it serves any purpose other than to keep the particles apart. It has, however, the advantage, in practice, that it overcomes the difficulty of ensuring that a direct current straight up the tube is uniformly distributed over the cross-section of the tube, and it has the further advantage that it gives more opportunity for the particles to slide over each other instead of colliding. This mechanical advantage is quite important; for a particle which is just (but only just) able to rise in a vertically upward current is constantly colliding with falling particles and driven down. When it is subjected to a helical upward

motion it escapes vertical collisions and can more easily work its way to the surface.

The benefit of the admission of the water to a Draper tubulure in a tangential direction is therefore apparent.

Classifiers of the Hindered-Settling Type.—None of the upward-current washers which have been described are of the true hindered-settling type. They depend essentially upon the free fall of particles in water, though it is true that the separation which they effect is influenced to some extent by the factors operative during hindered settling. No truly hindered-settling classifier has been used for coal cleaning, though there is no reason to anticipate that such an appliance would be otherwise than successful in cleaning coal. Indeed, in some respects, an advantage would be expected.

The phenomenon of hindered settling has been defined by Richards and Locke ("Text-book of Ore Dressing," New York, 1925, p. 127), as follows: "Hindered settling takes place where particles of mixed sizes, shapes and gravities in a crowded mass, yet free to move among themselves, are sorted in a rising current of water, the velocity of which is much less than the free-falling velocity of the particles, but yet fast enough so that the particles are in motion." "Free settling," according to the same authors, "takes place where individual particles fall freely, either in still water or against an opposing current, without being hindered by other particles."

It is difficult to specify exactly when practical conditions more nearly resemble those of free than those of hindered settling. Thus, in a conical classifier of any type, and especially a conical appliance like the Robinson coal washer, it is not possible to say that the conditions of separation of coal from dirt are entirely those of free-falling, yet they are certainly not those which Richards implies in his definition of hindered settling. By the definition, it is required that the mechanical reactions between the mass of particles shall be such that they stratify under the influence of an upward current which could not by itself cause stratification. The separation of light from heavy particles must therefore be effected by virtue of impulses and impacts between the particles, and must depend upon the effects of friction as well as upon the laws of fall of particles in liquids.

All the classifiers and washers that have been described in this section (upward-current separators) depend for their action, in the main, upon the free-falling properties of particles, and not on effects of hindered settling. There are, however, washers of the hindered-settling type, in which the particles separate partly by virtue of their free-falling properties, but mainly by reason of their behaviour when in a state of semi-suspension in a mass of particles of different dimensions. A classifier of this type was described by Richards (*Trans.* Amer. I.M.E., 1909, 41, 396), and its action may be explained

by reference to Fig. 75 (reproduced from Simons, "Ore-Dressing, Principles and Practice," New York, 1924.)

The classifying chamber consists of an irregular conical vessel with a cylindrical pipe attached to its lower and narrower end. In the upper part of the chamber, separation of the feed into two portions takes place, the lighter particles being supported by an upward current of water supplied through a series of holes in the hydraulic chamber, H, and overflowing at the discharge point. The remainder of the particles fall into the sorting column, C. The sorting chamber consists of two portions. In the upper portion, T, the sides are vertical, and in this, the " teeter " chamber, the particles are more or less in suspension, being too heavy to be lifted over into the pocket or settling chamber, but too light to fall against the inflowing water into the lower portion. The lower portion of the sorting column, K, has sides inclined to the vertical so that the passage of material downward is restricted and the velocity of the water is greater.

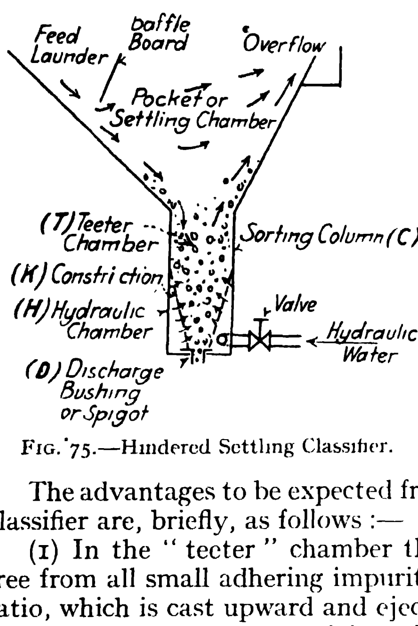

FIG. 75.—Hindered Settling Classifier.

In order that a particle may reach the discharge bushing or spigot, D, it must not only be capable of falling against the current in the settling chamber, according to the normal laws of fall, but must also pass through the suspension in the " teeter " chamber, and must then make its way through the bed of material in the constricted zone.

The advantages to be expected from the use of a hindered-settling classifier are, briefly, as follows :—

(1) In the " teeter " chamber the heavy particles are scrubbed free from all small adhering impurity below the critical size-density ratio, which is cast upward and ejected in the current of water.

(2) The suspended material results in an effective increase in the specific gravity of the medium in which the particles are moving, which increases the size ratio of equal falling particles and enables a wider range of particles to be separated. In particular, it prevents the particles of the lighter material from sinking, when otherwise they might be removed with the heavier material.

(3) Because of the mechanical agitation in the " teeter " chamber, a particle made up partly of one class of material, partly of the other, is liable to be broken up into two or more separate and " purer " particles. This latter effect would be a marked advantage if a relatively friable particle, such as a coal particle with shale attaching to it, were subjected to this method of dressing.

CHAPTER XII

THE THEORY OF ALLUVIATION APPLIED TO COAL CLEANING

THE principle of alluviation is most familiar in geological studies of the deposition of water-born solids. It is by alluviation that the bed of a river may silt up. Particles which are denuded from the banks or bed of a river are borne in the water current until, at some position where the speed of the current is reduced, the current is no longer able to bear them forward and they are deposited.

In a natural stream, the bed of the stream is composed, usually, of miscellaneous stones of different sizes and of different specific gravities. Because of the irregularity of the bed, the water flowing over it is turbulent, and if the individual stones are small enough to be affected by the agitation produced, they arrange themselves so that the smaller stones cover the larger ones. An exactly similar effect is produced on a pebbly shore where the agitation of the flowing and ebbing tides tends to accumulate a layer of tiny pebbles on the larger ones. In either of these cases, if the particles were of the same specific gravity, they would arrange themselves in layers with the smallest on the top. If, however, the particles were all of the same size but of different specific gravities, the layers would be arranged according to density with the lightest material uppermost. In a heterogeneous collection of particles, the tendency is to accumulate the smallest, lightest particles in the top layer and the largest, heaviest particles in the bottom layer. Between the two layers the particles arrange themselves so that there is a compensation between size and density, and if the sizes are not too widely apart, the lighter materials will segregate towards the top of the bed, the heavier towards the bottom.

Similarly, in a trough washer, in which a stream of water flows over an inclined surface in and among a multitude of coal and dirt particles, the water current does not maintain a streamline form, but is essentially turbulent. Its turbulent flow sorts out the particles in the manner previously considered, with the result that there is a tendency for the dirt particles to concentrate at the bottom and nearest to the plane. The lighter coal, on the other hand, tends to seek the uppermost layer. Between the top layer and the bottom layer will be a mixture of the larger coal and the smaller dirt. The layers of coal and dirt can then be separated by using a water current of suitable speed. The coal in the upper layers can be made to travel forward in the water current, whereas the dirt particles in the lower layers are not so free to move and can be made to remain stationary on the surface.

Early forms of trough washer depended upon alluviation only. The coal was washed down the trough by a current of water and, periodically, the feed was discontinued and the dirt on the bottom of the trough was shovelled away. In later forms of trough washer, modifications were introduced whereby the dirt could be removed continuously, and better methods of controlling the stratification were also devised.

Coal-cleaning appliances depending mainly upon the principle of alluviation may be grouped into four classes.

(1) Trough washers, in which coal and dirt are separated by a current of water flowing down a plane.

(2) Endless-belt washers, which are similar to trough washers in principle, but in which the belt surface is in continuous motion.

(3) Rheolaveur washers, which are essentially trough washers with different methods of discharging the products, and with a slight difference in principle.

(4) Concentrating tables, on which the separation effected by a current of water flowing down a riffled plane is facilitated by mechanical means.

This classification is not entirely satisfactory. In a simple trough washer and a Rheolaveur washer stratification is produced entirely by alluviation. There is, however, the difference that, in a simple trough washer, stratification is also induced by a current of water flowing among a multitude of particles (as in the bed of a natural stream), but in a Rheolaveur washer the particles are deposited from a rapidly moving current which is retarded by progressively reducing the inclination of the trough. In the Rheolaveur washer, therefore, the stratification more nearly resembles the silting up of a river bed and the deposition of solid particles in estuaries and at river-bends.

On a concentrating table, the principle of alluviation is also operative in causing stratification, but the mechanical agitation of the surface of the table probably assists the process. The principal function of the agitation, however, is to transport the lower layer of dirt to the position of its discharge.

It is common to all the four classes indicated that the particles in the layer of coal are carried to their discharge point by the water current. The four classes differ, however, in the means of removing the dirt.

There are several washers which do not fall strictly into any one of the above four classes. Thus the Murton and Blackett washers, though usually classed as trough washers, might with some justification be included as endless-belt washers. Similarly, several endless-belt washers differ little from shaking tables in so far as their design includes the principle of reciprocating movement in addition to that of continuous uni-directional movement. Nevertheless, the classification suggested is comprehensive, and simplifies the consideration of the various cleaning appliances which introduce alluviation.

It should be clearly understood that the principles of the separation of coal from dirt in jig and upward-current washers, which were discussed in Chapters III and IV, are not applicable to the separation of coal from dirt by a current of water flowing down an inclined plane. In jigs and upward-current washers the motion of particles in still water and in vertical currents of water is involved. In inclined-plane washers, the particles are not always free to " settle," the water currents are not vertical, and the phenomenon of friction is of considerably greater importance.

The principle of separation may be shown in the following way. Suppose that two cubical particles, A and B, of equal mass, m, but of different densities, s_1 and s_2, be at rest on a plane inclined at an angle α to the horizontal, the frictional force between each particle and the plane being greater than the force of gravity tending to cause them to slide down the plane.

Since the particles are of different material, the coefficients of friction between them and the plane will be different. But, assuming for the moment that they have the same coefficients of friction with the surface of the plane, μ, because their masses are equal, the frictional resistance to motion will be the same for each. Now, suppose that a current of water flows down the plane, the stream of water being of sufficient depth to cover each particle. The force exerted by the water current on each particle is proportional to the area of the face of the cube upon which the water impinges, *i.e.*, is proportional to $\left(\frac{m}{s_1}\right)^{\frac{2}{3}}$ and $\left(\frac{m}{s_2}\right)^{\frac{2}{3}}$, the length of each side of the cubes being respectively $\left(\frac{m}{s_1}\right)^{\frac{1}{3}}$ and $\left(\frac{m}{s_2}\right)^{\frac{1}{3}}$.

If s_2 is greater than s_1, the pressure of the water on the particle of specific gravity, s_1, will be greater than the water pressure on the particle of specific gravity, s_2. By adjusting the speed of the water current and the inclination of the plane, the conditions can be made such that the extra pressure on the particle of specific gravity, s_1, can cause it to move down the plane, leaving the particle of specific gravity, s_2, stationary.

The only opposition to the motion of the two particles is the frictional resistance of the plane and the same amount of resistance is offered to each. The calculation, therefore, shows that it is possible to separate two particles of different specific gravities when the same opposition to their motion is provided. Actually, in practice, other factors are brought into play, as a result of which the opposition to the forward movement of the heavy particles is greater than the resistance experienced by the light particles, and, simultaneously, a greater pressure is exerted on the light particles than on the heavy particles.

The calculation made does not particularise as to the sizes and

specific gravities of the particles, and the dependence of their motion upon their physical characteristics.

The conditions for separation can be more nearly defined by considering the general equation of motion of a particle made to move down an inclined plane by a current of water flowing down it. Adopting the notation previously employed, the equation may be written :—

$$\frac{ar^3s}{g} \cdot \frac{dv}{dt} = kbr^2(W - v)^2 - ar^3(s - 1)\mu \cos \alpha + ar^3(s - 1) \sin \alpha \quad (38)$$

W being the speed of the water current, v that of the particle, and k, a and b constants defining the shape of the particle.* Under the conditions of coal-washing practice, the particle does not slide down the plane in the absence of the water current. Hence the frictional term $ar^3(s - 1)\mu \cos \alpha$ is greater than the gravitational term $ar^3(s - 1) \sin \alpha$.

By simplification and integration between the limits $v = 0$ and $v = v$, the equation becomes :—

$$t = \frac{1}{2BA^2D} \log \frac{W - v + D}{W - v - D} \quad . \quad . \quad . \quad . \quad (39)$$

$$\text{where } B = \frac{g(s - 1)}{s}, \; A^2 = \frac{kb}{ar(s - 1)},$$

$$\text{and } D = \frac{\sqrt{\mu \cos \alpha - \sin \alpha}}{A}$$

The ultimate velocity of the particle may be obtained by making t infinite.

When $t = \infty$, $W - v - D = 0$, A, B and D being constant for a given set of conditions. If $W - v - D = 0$,

$$\begin{aligned} v &= W - D \\ &= W - \frac{1}{A}\sqrt{\mu \cos \alpha - \sin \alpha} \\ &= W - V\sqrt{\mu \cos \alpha - \sin \alpha} \quad . \quad . \quad . \quad . \quad (40) \end{aligned}$$

V being the terminal velocity of fall of the particle in still water.† This is a general equation applicable to any particle on a given plane with a given current of water flowing down it. The only variables for different particles are V and μ.

Applying the equation to the separation of coal from dirt, it follows that two particles, one of coal and one of dirt, will ultimately move with the same velocity if—

$$W - V_1\sqrt{\mu_1 \cos \alpha - \sin \alpha} = W - V_2\sqrt{\mu_2 \cos \alpha - \sin \alpha},$$

* This equation may be compared with that given in Chapter III for the motion of a particle on a plane with a current of water flowing *up* the plane.

† See Chapter III.

where V_1 and μ_1 relate to coal, V_2 and μ_2 to dirt. The limiting conditions for separation are, therefore, given when—

$$V_1\sqrt{\mu_1 \cos\alpha - \sin\alpha} = V_2\sqrt{\mu_2 \cos\alpha - \sin\alpha},$$

or, $$\sqrt{r_1(s_1 - 1)}\sqrt{\mu_1 - \tan\alpha} = \sqrt{r_2(s_2 - 1)}\sqrt{\mu_2 - \tan\alpha},$$

or, $$\frac{r_1}{r_2} = \frac{(s_2 - 1)(\mu_2 - \tan\alpha)}{(s_1 - 1)(\mu_1 - \tan\alpha)} \quad . \quad . \quad . \quad . \quad . \quad (41)$$

This relation between r_1 and r_2 defines the conditions under which two particles of different size ultimately move together. Under some conditions of coal washing on an inclined plane, the coal particle is made to move down the plane and the dirt particle is not allowed to acquire a motion down the plane, but is made to remain stationary relative to the separating surface. In other words, the acceleration of the coal particle given by equation 38 is positive, and that of the dirt particle is zero or negative. The limiting conditions for separation are then obtained by considering the size of two particles each with zero acceleration. The coal particle is then just incapable of being moved by the current, the dirt particle is just on the point of being moved. Under these conditions, two equations are obtained, namely :—

For coal, since $v = o$ and $\frac{dv}{dt} = o$,

$$kbr_1^2(W)^2 - ar_1^3(s_1 - 1)\mu_1 \cos\alpha + ar_1^3(s_1 - 1)\sin\alpha = o \quad . \quad (42)$$

And for dirt,

$$kbr_2^2(W)^2 - ar_2^3(s_2 - 1)\mu_2 \cos\alpha + ar_2^3(s_2 - 1)\sin\alpha = o \quad . \quad (43)$$

Dividing each side of equation (42) by r_1^2 and each side of equation (43) by r_2^2 and equating,

$$kbW^2 - ar_1(s_1 - 1)\mu_1 \cos\alpha + ar_1(s_1 - 1)\sin\alpha$$
$$= kbW^2 - ar_2(s_2 - 1)\mu_2 \cos\alpha + ar_2(s_2 - 1)\sin\alpha,$$

whence,

$$r_1(s_1 - 1)(\sin\alpha - \mu_1 \cos\alpha) = r_2(s_2 - 1)(\sin\alpha - \mu_2 \cos\alpha),$$

which leads to the same size relationship as in the case of separation by taking advantage of the ultimate velocity of the particle, namely

$$\frac{r_1}{r_2} = \frac{(s_2 - 1)(\mu_2 - \tan\alpha)}{(s_1 - 1)(\mu_1 - \tan\alpha)} \quad . \quad . \quad . \quad . \quad . \quad (44)$$

This elementary examination of the motion of two particles on a plane down which a current of water is flowing is not directly applicable to the conditions arising in coal-cleaning practice. All that it indicates is that a separation of two particles of different specific gravities can be effected, and by comparison with the size relationship for separation in an upward-current washer, namely,

$$\frac{r_1}{r_2} = \frac{s_2 - 1}{s_1 - 1}$$

it appears that the two particles can have a wider range of size than if they were to be separated by an upward current. The coefficient of friction between shale and steel (or wood) is greater than that between coal and steel (or wood), and $\mu_2 - \tan\alpha$ is therefore greater than $\mu_1 - \tan\alpha$, making the term $\frac{\mu_2 - \tan\alpha}{\mu_1 - \tan\alpha}$ greater than unity.

The factors which are brought into play in practice, but which have been excluded from consideration hitherto, all tend to widen the range of sizes. For this reason, the ratio of the sizes of coal and dirt that can be separated by a trough washer in practical operation, is greater than the ratio that can be separated by the simple action of a current upon two isolated particles. Consequently, in this respect, trough washers have a greater advantage over upward-current washers than simple calculation suggests.

The ratio of sizes is not, of necessity, greater, or even as great as the ratio for separation in a jig, because in a jig the velocity during the initial phase of fall in still water is a factor, as well as the terminal velocity of fall, and the size ratios for these two phases of vertical fall in water are not identical. Theoretically (equations 41 and 44) the size ratio should be the same on an inclined plane whether the initial or ultimate phases of the motion of the particles is considered.

When a water current on an inclined plane is utilised for separating raw coal into coal and dirt, the mass effect must be taken into account, and it is not satisfactory to consider that each particle, whether of coal or shale, is in contact with the plane. As has been stated, the particles tend to stratify with the coal in a layer above the dirt.

The tendency for stratification in a trough washer cannot easily be examined mathematically because of the number and variety of assumptions that must be made. It is known, however, that this stratification in a simple trough washer is a gradual process. As an indication of its gradual nature it may be mentioned that an early trough washer erected at Ince, near Wigan, which is described in Chapter XIII, was 600 ft. long, 400 ft. being used as the washing section.

In the first portion of a trough washer, the only definite separation that has taken place is that some of the large heavy shale particles have taken up a position underneath the remainder of the particles and some of the small light coal particles are uppermost. Between the layers of small coal and large shale, a layer of assorted particles is present.

As the particles travel further down the trough, the particles forming the intermediate layer tend to arrange themselves into a further series of layers, in which the lighter particles are uppermost. The water current is always tending to move the lighter particles forward and to cause them to leave the heavier particles behind,

because, as has already been shown, the pressure of the water on the lighter particles is greater for a given mass than the pressure on a denser particle.

After some distance of travel the classification of the raw coal is fairly complete. The smallest coal is uppermost and the largest dirt is in contact with the plane, but because of the restricted freedom of movement in the crowded mass of particles, some of the lighter coal particles are mechanically entangled in the deposited dirt, and can only be released by the operation of an additional disturbing force.

The influence of a number of factors upon the segregated mass must now be taken into account They may be considered under three headings, namely :--

(*a*) The shape of the particles,
(*b*) The strength of the water current,
(*c*) The frictional resistance to motion in different parts of the bed.

(*a*) *The Shape of the Particles.*—Shale particles are, generally, of a flatter shape than coal particles and, in a trough washer, they tend to settle or slide with their largest face underneath, consequently, for a given mass, they expose a smaller area of cross section to the pressure of the current than do the more regularly shaped coal particles.

Suppose, as a hypothetical case, that we consider a cubical coal particle and a particle of shale composed of two smaller cubes joined at one face (a rectangular parallelopiped) And suppose that the two particles have the same weight, the specific gravity of the shale being twice that of the coal. Each particle will tend to present its smallest face to the current, so that, on the coal, the pressure of the current will be exerted over a surface area of r_1^2, r_1 being the length of one side of the cube. The pressure of the water being assumed to be proportional to the area of the face exposed to it, the force on the coal particle will be kr_1^2, k being a constant. The force per unit mass will be $\frac{kr_1^2}{s_1r_1^3} = \frac{k}{s_1r_1}$, s_1 being the specific gravity of the coal.

The volume of the shale particle is $r_2^2 \times 2r_2 = 2r_2^3$ and its weight being equal to that of the coal particle, $2r_2^3s_2 = r_1^3s_1$. As by hypothesis, $s_2 = 2s_1$, $\frac{r_2}{r_1} = \left(\frac{1}{4}\right)^{\frac{1}{3}}$. The force exerted by the water current will therefore be equal to $kr_2^2 = kr_1^2\left(\frac{1}{4}\right)^{\frac{2}{3}}$, or less than one-half of that exerted on a coal particle of equal weight.

Taking their coefficients of friction as equal (actually that of the shale is the greater), the frictional resistance to each will be the

same, because their masses are equal. The influence of shape is therefore directed towards a facilitation of the separation of coal from dirt.

(*b*) *The Strength of the Current.*—As the current flows down the plane, its speed is slower near the surface of the plane than in its upper layers. If a layer of large heavy shale particles rests on the surface of the plane, the layers of water near to the plane are still further retarded because of the frictional resistance of the surface of the particles. The upper layers of the water do not meet so much extra resistance, because the particles towards the top of the bed are more loosely packed. The conditions are, therefore, more favourable to the movement of the coal, which preponderates in the upper layers of the bed, and unfavourable to the movement of the shale in the lower layers.

If a cubical particle of coal is resting on a bed of shale and is met by a relatively strong current of water at its upper edge and a weaker current at its lower edge, the strong current at the top will tend to tilt the particle and make it roll along on the top of the shale. Consequently there is less frictional resistance to its motion than if it were to slide over the shale. There is much less (if any) tendency for the flat shale to roll along in this manner and the unevenness of the water current thus tends to reduce the frictional opposition to the coal without proportionally reducing the frictional opposition to the shale.

Reverting to the conception of three layers of material, the top one of coal, the middle one of larger coal and small dirt, and the lowest one of large dirt, it may be seen that there will be an opportunity for the small dirt to fall out of the middle layer into the interstices between the large dirt particles in the bottom layer. This tendency will arise because the turbulent water in the middle-depth of the stream is being constantly checked on account of the uneven bed of dirt on and in which it flows, and the temporary and relatively quiescent conditions that then arise will afford an opportunity for the small heavy particles to sink into the gaps in the bed of large heavy dirt particles, the coal particles being prevented from so sinking on account of their greater size.

Except for these considerations, the fact that the water is turbulent does not interfere with the theoretical deduction of the limiting degree of separation possible, for despite the fact that the water does not impinge on the surface of the particles normally, but in a variety of directions, nevertheless, if all the variety of currents be resolved in a direction parallel to the plane, some mean value is obtained which is the same for all particles.

(*c*) *The Frictional Resistance to Motion.*—The lack of uniformity in the speed of the water current has already been shown to provide a tendency to decrease the frictional resistance to the coal particles in the upper layers of the bed of material, without correspondingly reducing the frictional resistance to the movement of the shale

particles in the lower layers, this effect being produced by imparting a rolling motion to the coal particles.

The frictional resistance to the motion of the shale is also increased because the shale is pressed on the surface of the plane by the weight of the coal above it. At the same time, the tendency for coal particles to be opposed by friction with the plane is reduced because coal particles rarely come into direct contact with it. If they do, they will move down the plane more rapidly than do the shale particles and, on colliding with a flat shale particle, will probably be lifted from the plane if a favourable opportunity (a loose bed above them) occurs.

The calculations at the beginning of this chapter showed that there was a tendency for coal particles to be moved away from shale particles by a current of water when each was opposed by the same frictional resistance. The ratio of separable sizes in these circumstances was found to be

$$\frac{r_1}{r_2} = \frac{(s_2 - 1)(\mu_2 - \tan \alpha)}{(s_1 - 1)(\mu_1 - \tan \alpha)}$$

In the circumstances of coal-cleaning on an inclined plane, the frictional opposition to motion is unequal, the shale being opposed by forces of greater magnitude. The sizing ratio is therefore increased to a value greater than that suggested by the formula, and this is known to be the case in a well-designed trough washer, such as the Rheolaveur.

The theory outlined explains the principles upon which coal is separated from dirt by the simple flow of a current of water over an inclined plane and is applicable directly to trough washers. In a trough washer, the coal is carried forward by the stream of water, and the dirt settles to and remains on the bottom of the trough. In endless-belt appliances, the coal passes down the belt in the current of water. The dirt is held to the surface of the belt by frictional forces and travels with it in the direction opposite to that of the coal.

The action of a Rheolaveur washer is similar to that of a trough washer, though it differs in certain respects. The coal and dirt are given initial velocities, and the actual separation is effected by allowing the coal particles to pass forward whilst the dirt particles are retarded.

The cleaning of coal on concentrating tables also depends upon principles similar to those of trough-washing, but the reciprocating motion of the surface of the table and the presence of riffles upon it introduce modifications which require to be dealt with separately.

CHAPTER XIII

TROUGH WASHERS

TROUGH washers were the most primitive of all forms of washer used for the enrichment of metalliferous ores, but they were not used for the cleaning of coal until attempts had first been made with various forms of jig. As already stated in Chapter V, trough washers were first employed in France and Belgium for coal washing about 1841. In Germany, where the first attempts at coal washing were made, various forms of jig were used, but trough washers apparently received little consideration. The recorded evidence shows that in Great Britain trough-washing methods were not employed until Bérard's continuous coal-washing jig had been introduced in 1849. The earliest recorded trough coal washer used in Great Britain, so far as the authors are aware, was that of Green and Bell at Shincliff Colliery, Durham, in 1857 (J. A Birbeck, *Proc.* Chest. and Derby Inst. Eng., 1872, 1, 76), described in Chapter V. This trough washer, however, involved the use of various mechanical devices for stirring and aiding the cleaning out of the settled refuse from the troughs, so that it would appear likely that simpler forms of troughs were in use at a somewhat earlier date.

After the possibility of cleaning small coal had been demonstrated, trough coal washers were adopted in Durham, Yorkshire and Lancashire, and because of the simplicity of their construction and working, and the difficulties which were encountered in the early forms of jig washer, they became very popular for the cleaning of fine coal for coke manufacture.

Early Types of Trough Washer.—Some of the earliest forms of troughs or "spouts" were extremely long, for besides their use for coal-cleaning purposes, they also served to convey coal from one point to another. Thus at Ince, near Wigan, Lancs. (according to G. Gilroy, *Trans.* N.E. Inst. Min. Eng., 1865-66, 15, 61), the total length of the trough erected in 1864, was 600 ft. The first 275 ft. of its length had a fall of 1 in 18, followed by a section 40 ft. in length, with a gentler inclination of 1 in 24; the trough in both these sections was 10 in. wide and 10 in. deep. After the second, there followed a further section 70 ft. in length, with the same inclination as the second section (1 in 24), but the trough was wider, namely, 23 in., and had sides of the same height (10 in.). This first 400 ft. length was the true washing portion. The remainder of the length of the trough (200 ft.) was used to convey the washed slack to the beehive coke ovens.

In the washing section of the trough, the slack was carried along by the stream of water, and the refuse was gradually deposited over the whole length. The heavier dirt settled in the first section. The lighter dirt tended to settle in the second section, where the gradient was less, and in the third and wider section, where the water current was slower. Midway in this portion of the washing trough was a dam 3 in. high, which arrested the progress of the lighter shale particles. In operation, when the settled refuse had attained the level of the top of the dam, a second dam, also 3 in. in height, was added to the first. The segregation of the refuse was further aided by additional dams, fixed at distances of 24 ft. and 55 ft. from the first dam. In the third and wider section of the trough, several valves were fitted for the removal of the refuse. The first valve was at a point 15 ft. before the first dam, and additional valves were fitted in front of the two other dams. When the settled lighter dirt had risen to the top of the first 6 in. dam, the first refuse valve was opened (the coal supply being cut off) and the refuse was flushed from the trough into wagons. The settled refuse from the succeeding length of the trough up to the third dam was similarly removed through valves. All the valves controlling the working of the washer were under the control of a man at the coke-oven end of the trough.

Fine coal was admitted from a hopper at the colliery screening plant, together with a supply of water, and the washed coal was carried along by the stream of water over the series of dams to the last section of the trough, which conveyed it to the hoppers at the beehive-oven plant. The capacity of the washer was said to be 12 tons of coal per hour, about 6 per cent. of refuse being removed. 80 to 90 gallons of water were circulated per minute or 450 gallons per ton of coal washed. The labour costs were stated to be 0·3*d*. per ton of slack.

The washing troughs were not invariably of great length, and sometimes quite short troughs were used, as in ore-dressing practice. Thus at South Tyne Colliery, Haltwhistle, Durham (according to J. Brogden, *Trans*. S. Wales Inst. Eng., 1876–77, 10, 119), the first trough used was a single length of 25 ft. At the same colliery several troughs were later used for coal washing which were 29 in. wide and 6 in. high; the length, unfortunately, is not stated. The bottom and sides of the trough were grooved at intervals to receive dams, which were inserted during the process of washing. Coal was shovelled into the trough by hand and a man assisted the spreading of the coal by means of a rake. This method of procedure is very similar to the methods used in ore-dressing practice, in appliances called "buddles."

At Aldewarke Main, Yorkshire (according to the report of Scottish Coal-Cleaning Committee, *Trans*. Min. Inst. Scot., 1889–90 11, 168), double troughs, each 16 in. wide and 9 in. high, were employed, with a fall of 1 in 24. A length of 50 yd. was considered satisfactory for washing 8⅓ tons per hour. The length of the troughs

employed was much greater than 50 yd., but this was because they were used also to convey the washed coal to the coke ovens, situated

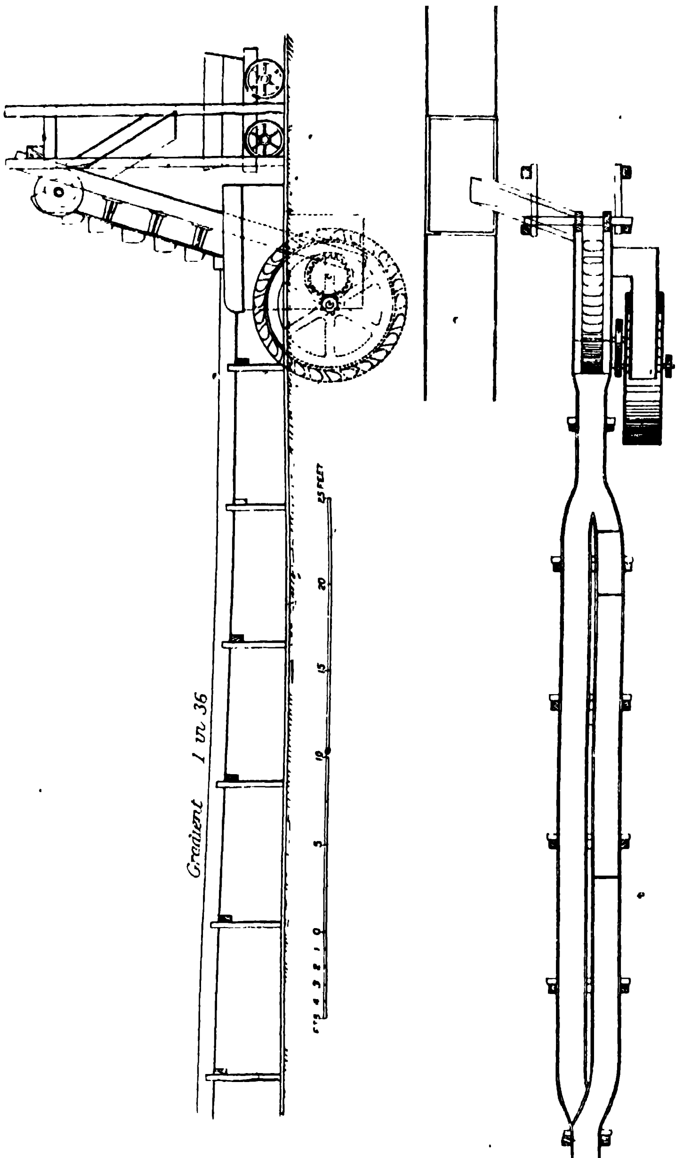

Fig. 76.—Simple Trough Washer, Flimby Colliery (1881).

some distance from the colliery. The bottoms of the troughs were covered with glass plates to minimise the wear, and dams were fitted at short intervals to aid the control of the washing process. The unwashed coal contained 12 per cent. of free dirt, which was

reduced to 4 per cent. by washing. The coke produced from the washed coal contained only 6 to 7 per cent. of ash. The pea nuts were washed separately in troughs, and other sizes of coal were crushed by rollers before being delivered to the troughs or washing "spouts." The water from the troughs was run into one of three settling tanks, each 48 ft. long by 20 ft. wide and 7 ft. deep, and capable of holding 40 tons of fine dirt. Part of the water was re-used after settling, but some of it ran to waste. The cost of elevating, crushing and washing was stated to be 2·75*d*. per ton of coal.

At Nunnery Colliery, Yorkshire (according to the same authority), the troughs were of similar dimensions to those at Aldewarke, and in a length of only 54 ft. it was found possible to wash 15·7 tons of coal per hour. The actual washing length of trough was succeeded by a further trough to convey the coal about half a mile to the ovens. (Hence it used to be claimed, with pride, that Nunnery possessed the longest washer in the country.) Part of this trough was placed horizontally to enable the lighter portions of dirt to settle. In a period of one day (14 hours) about 4 tons of light dirt were collected in this horizontal section and were flushed out at night time. The amount of refuse removed from the coal is stated to be about 10 per cent., but the ash contents of the raw and washed coal are not recorded. Pit water was used, amounting to 200 gallons per minute (or 750 gallons per ton of coal), and was run to waste after use. The cost of cleaning is stated to be 1·4*d*. per ton of coal.

At Flimby, Cumberland, two parallel trough or "rhone" washers, illustrated in Fig. 76, were employed. They were 150 ft. long, 17 in. wide and 13 in. deep, the inclination being 1 in 36. The troughs were supported from the ground by a series of trestles. In each trough dams 2 in. high were fitted 20 ft. from the lower end. The two troughs were used alternately, one trough being cleaned out whilst the other was in operation. The washed coal overflowed into a tank from which it was elevated to a hopper by a bucket elevator operated by a 6 ft. water wheel, which was worked by the water overflowing from the tank (Fig. 76). The percentage of dirt removed was 18 per cent., and the cost of washing was 1·18*d*. per ton of coal.

On account of the simplicity of their construction and the ease with which they could fill a dual *rôle* of washer and conveyor, trough washers were frequently employed in the coalfields where coking coal was mined, to prepare the coal for beehive coke manufacture. By their use a clean coal was readily obtained, but there was usually a loss of considerable quantities of coal in the refuse. For example, D. Cowan (*Trans*. Min. Inst. Scot., 1888–89, 10, 229) records the following figures for the working of a number of trough washers.

In the last example, the removal of dirt from the coal was not very efficient, but, to compensate for this, there was little loss of coal in the dirt. These results are typical of many analyses we

Place.	Type.	Per cent. Dirt in Coal. Before Washing.	Per cent. Dirt in Coal. After Washing.	Per cent. Coal in Refuse.
Carron, Scotland	Simple	14·32	4·45	9·14
Teesdale, Durham	Bell & Ramsay	13·84	3·40	13·38
E. Howle	Ramsay	25·35	13·51	trace

have made on the working of a simple trough washer which is still in use for the production of washed coal for the manufacture of beehive coke. According to these analyses, whilst an efficient cleaning was readily obtained, a loss of about 10 per cent. of coal in the refuse was a common experience.

The disadvantages of the simple trough, apart from the loss of coal in the refuse, lay in the high water consumption, and the difficulty and expense of labour. In many cases the water, after passing through the trough, was not re-used, and carried away with it a large portion of the finest coal (as well as of dirt), so that its disposal frequently proved to be a difficult matter. The very length of the trough in many cases made it difficult and costly to retain the water for re-use, and, moreover, its extreme length made it necessary to work in the open in all sorts of weather, thus throwing an additional difficulty in the way of control.

Once the coal and water supplies had been adjusted, the manual labour required for the control of a trough washer consisted in turning over the settled dirt with a shovel to permit the recovery of some of the coal which was trapped with the dirt. When the dirt had accumulated to a certain depth, the supplies of raw coal and water were cut off and the dirt was removed from the floor of the trough. Sometimes this involved the shovelling of the dirt into tubs, and in certain cases returning some of the refuse from the lower section of the trough to the feed end for re-washing. Continual attention had to be paid to stirring the settled dirt to prevent undue loss of coal in the refuse. Thus the work was arduous and, in wet weather, disagreeable.

The Green and Bell Trough Washer.—Many attempts have been made to increase the mechanical efficiency and throughput of simple trough washers. One of the first attempts was made in 1857 by Green and Bell, who introduced mechanical devices to stir up the settled dirt and aid the removal of the refuse from the trough. This washer is illustrated in Fig. 77, in which dams, *b*, are connected by bell cranks, *e*, to a connecting rod, *f*, by the adjustment of which the dams could be raised clear of the trough. A series of rakes, *c*, were actuated by the movement of a rod, *d*, to stir up the settled

dirt. When sufficient dirt had accumulated the rake and dams were raised, and the refuse flushed out through the door, *g*.

The Wunderlich Trough Washer.—A trough washer in which an attempt was made to reduce the loss of coal with the refuse was the Wunderlich, used in Bohemia (Waltl, *Oest. Zeit. Berg. Hütten.*, 1897, 45, 357). In this washer, two troughs, each 18 in. wide, were employed, the refuse from one being elevated to the head of the other for re-washing. Belts were employed for conveying the coal. The capacity of the trough, when washing sized coal, was found to be 15 tons per hour for coal of 1 to 2 in. size, and 9 tons per hour for coal of $\frac{1}{5}$ to $\frac{1}{2}$ in. size, with a water consumption of 740 gallons per minute, or 4,933 gallons per ton of fine coal. When the refuse from one trough was re-washed in the second trough, it was unnecessary to size the coal before washing. The cost was said to be 10*d*. per ton.

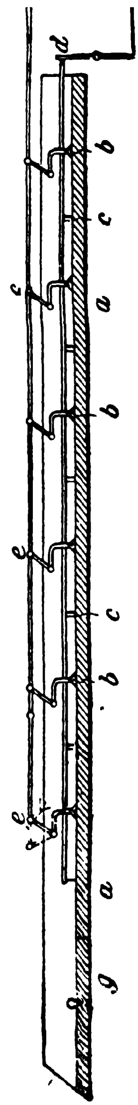

FIG. 77.—Green and Bell Trough Washer (1357).

The Bell and Ramsay Trough Washer.—The Bell and Ramsay washer, which was in use in 1889 (*Trans.* Min. Inst. Scot., 1889–90, 11, 186), was an improved form due to the mechanical ingenuity of Erskine Ramsay, who was responsible for mechanical improvements of the Robinson washers used in America (p. 194). The Bell and Ramsay system illustrated in Figs. 78 and 79 included devices for similar purposes to those in the earlier forms of Green and Bell. Stirring rods were used as well as a series of dams, the essential difference between this and the Green and Bell washer being the introduction of mechanical means to lower one end of the trough to flush out the settled refuse instead of raising the dams and scrapers. At Robin Hood Colliery, Maryport, Cumberland, the coal was tipped on to a jigging screen through which the fine coal passed into a pit whence it was elevated to the Bell and Ramsay washer. (The large coal was hand-picked on the screen and passed into the wagon illustrated in Fig. 78.) Each trough was made of sheet metal and was semicircular in cross-section. Both were hinged at the upper end, the lower end being supported by an arm connected to a hand lever for purposes of control (Fig. 78). A shaft, provided at frequent intervals with stirrers (C, Fig. 79), extended the whole length of each trough, and had a transverse oscillating motion. A

series of dams were also employed, these in the upper portion of the troughs being adjustable by means of levers (A, Fig. 79), and others placed in the lower part of each trough were fixed to framework above the trough, and so were permanently fixed in position when the trough was in operation. Fixed scum plates were also used in conjunction with several fixed dams (B, Fig. 79) to prevent any coal from floating away on the surface of the water.

In operation, the coal was carried by the water stream along the

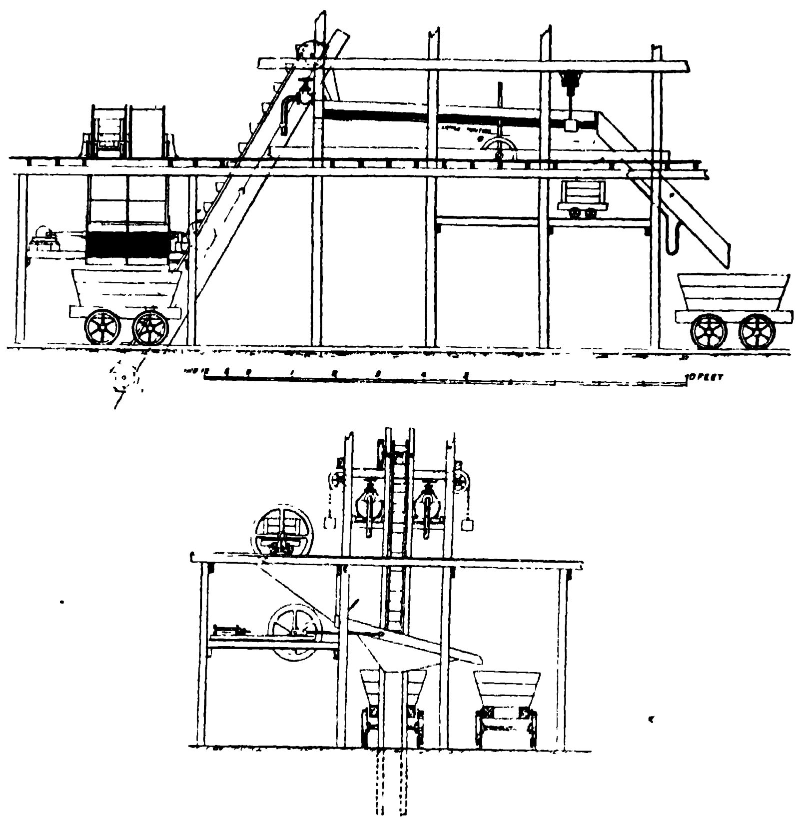

FIG. 78.—Bell and Ramsay Trough Washer.

trough, whereas the dirt settled to the bottom and was arrested by the dams; its agitation by the stirrers prevented excessive quantities of coal from being entrapped in the refuse. When a certain quantity of dirt had accumulated in one trough, the supply of raw coal was deflected to the other, and the lower end of the first trough was lowered clear of the fixed dams and stirrers by manipulating the hand lever A, Fig. 79. The adjustable dams of the upper portion of the trough were raised by the appropriate hand levers to enable the stream of water to flush out the refuse into a wagon (*vide* Fig. 78).

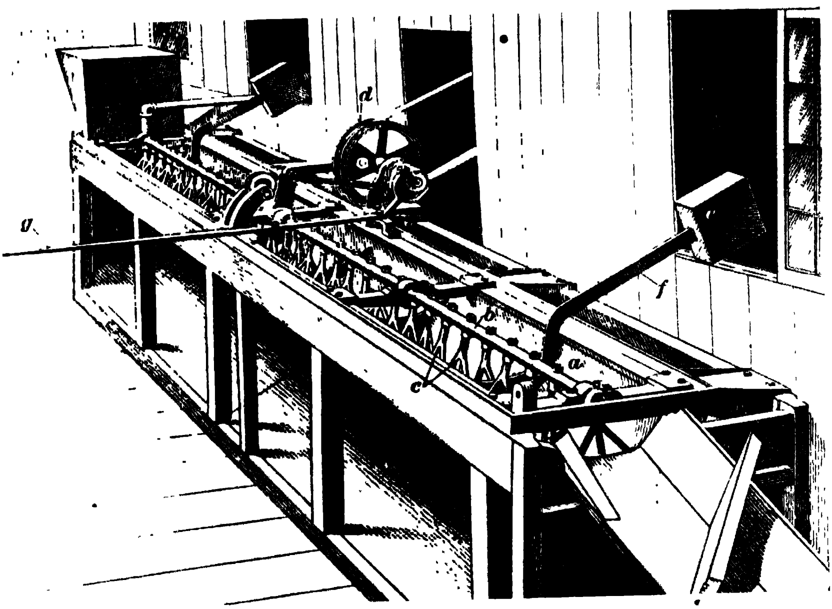

FIG. 80.—View of Scaife Trough Washer

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In this way the labour of controlling the working of the trough washer was much reduced and the throughput of coal increased.

The Bell and Kirkby trough washer (*Iron and Coal Trade Review*, 1904, 68, 1057) was an improved Bell and Ramsay washer in which the inclination of the trough was variable.

The Scaife Trough Washer.—A somewhat similar type of trough washer, the Scaife, has been used in America (Fulton, Coke, Scranton, Pa., 1905). The trough, illustrated in Figs. 80 and 81, was semicircular in cross-section and was 24 ft. long, and 2 ft. in diameter. It was hinged along one side to the supporting framework. The trough was retained in its washing position by the balance weight, *e*, and the lever, *f*, fixed to the trough; the trough was locked in position by means of a tongue on the operating lever, *g*, and an eye in the trough. A number of dams were fixed in the trough at intervals, and a series of stirrers, *c*, were fitted to a shaft, *b*, which was given a reciprocating movement, transverse to the length of the trough, by a crank and a connecting rod fitted to the driving pulley, *d*.

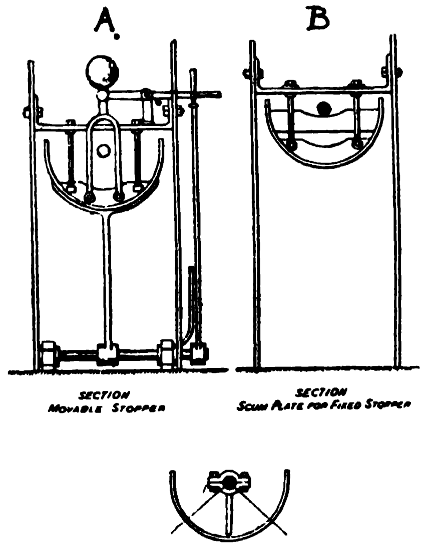

FIG. 79.—Bell and Ramsay Trough Washer: Sections through Troughs. (A) Movable Dams; (B) Fixed Dams and Scum Plates; (C) Stirrer.

In operation, coal was carried by a stream of water along the trough, which was suitably inclined, and the dirt settled to the bottom, where its movement was arrested by the dams. The settled refuse was agitated by the stirrers to avoid undue loss of coal in the rejected refuse. When a certain quantity of dirt had collected in the trough, the supply of coal and water was cut off and the operating lever, *g*, moved to unlock the fixing of the trough. The levers, *f*, were then moved over towards the trough. The trough was thus depressed and the refuse was discharged into a shoot. A further movement of the lever, *g*, brought a clutch into operation and the trough was raised to its washing position, with the aid of the chain, *h*, the weight, *i*, and the driving pulley, *d*. The trough was locked in position again, the clutch being at the same time thrown out of action, and the balance weights adjusted. The washer was then ready for re-use.

The McLellan Trough Washer.—The McLellan trough

washer (*Iron and Coal Trade Review*, 1904, 69, 413) consisted of two troughs on either side of a central trough ; the dirt was carried from the side to the central trough by arms working on shafts, and was discharged thence by means of a screw conveyor.

The Lodge Trough Washer.—At Lodge and Longrigg collieries, Slamannan, Scotland, simple troughs were used as early as 1869. At Lodge colliery, in order to increase the output, double troughs were built, 60 ft. long, 11 in. deep, and 4 ft. wide over the first portion of the length and 3 ft. wide over the remainder (Fig. 82).

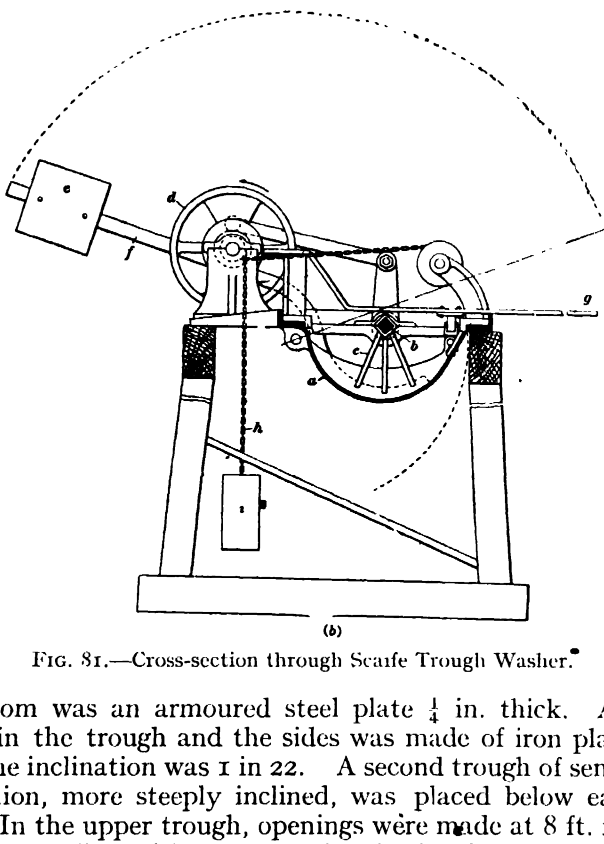

FIG. 81.—Cross-section through Scaife Trough Washer.

The bottom was an armoured steel plate $\frac{1}{4}$ in. thick. A central division in the trough and the sides was made of iron plate $\frac{3}{16}$ in. thick ; the inclination was 1 in 22. A second trough of semicircular cross-section, more steeply inclined, was placed below each main trough. In the upper trough, openings were made at 8 ft. intervals, and were supplied with stoppers fitted with handles, F, for their removal. Immediately in front of each opening a dam, C, was placed to aid the segregation of the refuse. In operation, each of the washing divisions was used alternately. When one was in use, the other was being cleaned out by raising the stoppers, D. The refuse was flushed into the lower trough to the point, E, and loaded into wagons. The washed coal overflowed into the shoot for delivery to the coke-oven plant.

At Longrigg Colliery the coal had a high carbon content (90 per cent.) and a high specific gravity. This made its separation from the accompanying shale a matter of greater difficulty than the washing of ordinary bituminous coal. The raw coal contained from 20 to 25 per cent. of dirt, and although it could be cleaned satisfactorily in simple trough washers, the yield and throughput were small. Various attempts were made to use "bash" washers, but in them the coal was subjected to too much disintegration. Accordingly, a modified trough washer was adopted (*Trans.* Min. Eng. Scot., 1889-90, 11, 217), in which mechanical means were introduced to increase the rate at which the settled refuse was removed and so to increase the throughput of coal. The trough was 60 ft. long, 2 ft. wide and 11 in. deep, and set at an inclination of 1 in 18. The coal was admitted, together with part of the water supply, from a shoot at a distance of 15 ft. from the upper end. The remainder of the water was introduced at the extreme upper end of the trough, through a cock. A series of dams or scrapers 2 in. high were set on a connecting rod inside the trough at intervals of 4 ft. The scrapers moved continuously up the trough, and returned underneath it, passing over drum heads at both ends. By this means the settled dirt was carried to the feed end, and, in the upper 15 ft. length, was re-washed by the main stream of water to recover some of the mechanically-entangled coal particles. The refuse was then discharged into wagons. The lighter coal particles were carried over the travelling dams and passed from the lower end of the trough into a de-watering rotating cylindrical sieve. This trough, which was described as self-cleaning, washed 20 tons of coal per hour.

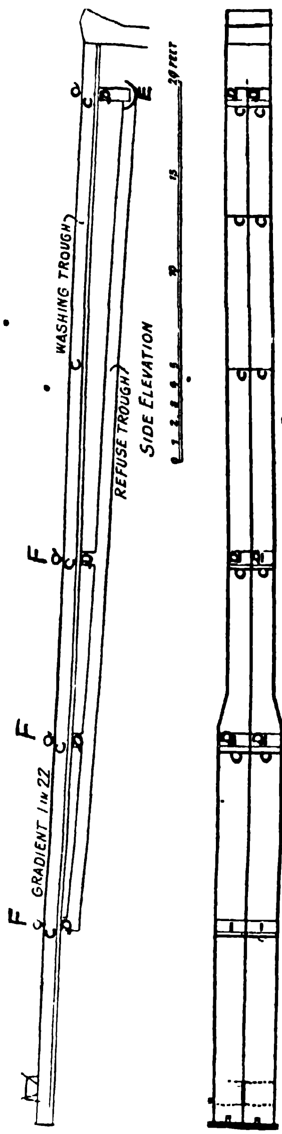

FIG. 82.—Lodge Trough Washer.

The Elliott Trough Washer.—In 1894, Elliott (*Colliery Guardian*, Nov. 16th, 1894) designed his well-known trough washer which was, and still is, built by the Hardy Patent Pick Co., Ltd., of

Sheffield. In many respects it was similar to the Longrigg washer previously described. Elliott followed the practice of admitting the raw coal at a point some distance from the upper end of the trough to allow the settled refuse to be re-washed before rejection. For this purpose he introduced the raw coal midway in the length of the trough, so that only half the total length was used to wash the coal the other half being utilised in re-washing the refuse. (It may be noted that, in ore-dressing practice on endless belts, this method had been in use for fifty years.)

An early form of the Elliott trough washer was built of cast iron or steel plate, and was 60 ft. in length with an inclination of 1 in 12. The bottom of the trough was flat and 18 in. wide, with sloping sides, so that the top width of the trough was 30 in., the depth being 11 in. Sprocket wheels were fixed at each end of the trough to carry an endless chain to which scrapers were fixed at intervals. These formed travelling dams which moved along the bottom of the trough towards the upper end. Coal, from which the fine dust had been removed, was delivered to the trough 25 to 30 ft. from the upper end. Water was admitted with the coal at the mid-length of the trough and a further supply entered at the upper end. The stream of water carried the coal down the trough in the coal-washing section, cascading over the scrapers to the lower end, where it passed over a perforated plate for de-watering. The dirt settled on to the floor of the trough and was carried to the upper section. From the upper end it passed into a dirt shoot for disposal.

Flat scrapers were at first used, but with them there was a tendency for dirt to be left at the sides of the trough at the edges of the advancing scrapers. Some of the dirt was therefore mixed with the coal carried down in the stream of water. Concave scrapers driven by single or double chains were therefore substituted, and, later, serrated or corrugated scrapers were devised (E. Greaves, *Trans.* Inst. Min. Eng., 1906–07, 33, 138). By the use of these modified scrapers, a better stirring-up of the settled dirt was effected, thus reducing the danger of excessive loss of coal with the refuse. In this way, and with the re-washing of the settled dirt, the efficiency of the Elliott trough washer was higher than that of the earlier simple trough washers. Moreover, in order further to improve the efficiency, Elliott trough washers were designed to treat different sizes of coal, the raw coal being divided into a number of fractions by means of a rotating cylindrical sieve. Each sized fraction was treated in a separate appliance, for which the scraper mechanism and water flow could be individually adjusted. In one instance coal was divided into three fractions, namely, $\frac{1}{8}$ to $\frac{1}{4}$ in., $\frac{1}{4}$ to $\frac{3}{4}$ in., and over $\frac{3}{4}$ in. size. For the larger sizes of nuts the scrapers were fixed at intervals at 4 to 5 ft., and were $2\frac{3}{4}$ in. high; for the smaller sizes of coal the scrapers were $1\frac{1}{2}$ to 2 in. high and were set 2 to 3 ft. apart. The speed of the chain scrapers was varied by differential head stocks. A trough dealing with the smallest sizes of coal could

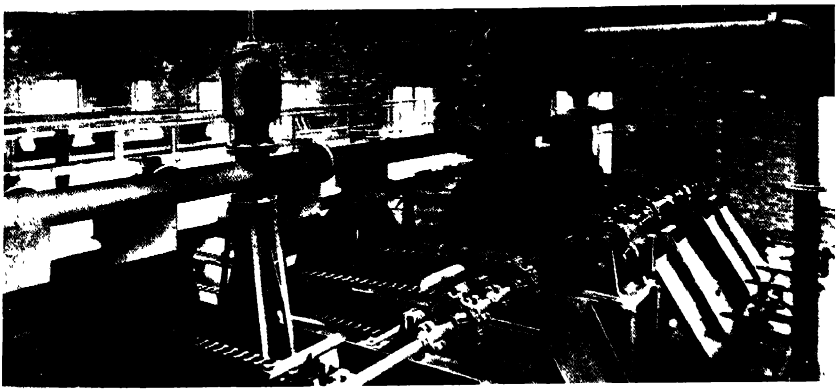

FIG. 83.—Battery of Elliott Washing Troughs with Single-chain Scrapers

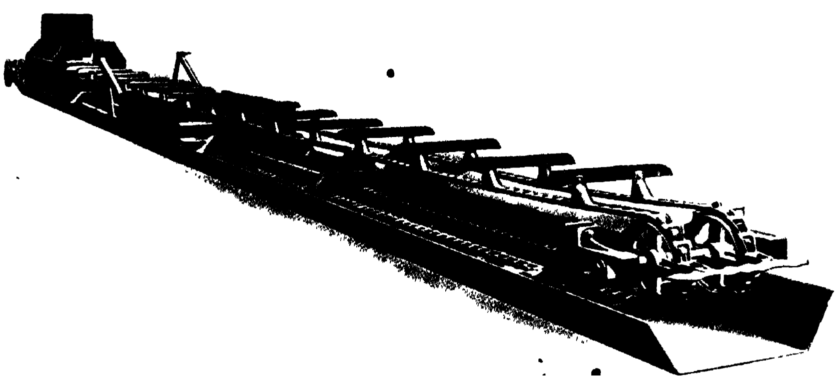

FIG. 84.—Elliott Trough with Double-chain Scrapers.

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treat about 4 tons of coal per hour, and, with the largest size, 12 tons per hour could be washed. A view of a battery of Elliott trough washers with single-chain scrapers is given in Fig. 83 and a single trough with double chain scrapers of corrugated section is shown in Fig. 84.

In more recent times the Elliott washer has been improved by the addition of a primary washer or table which enables the larger-sized dirt to be removed before the coal passes to the washing trough proper. A view of the primary washer is given in Fig. 85. The raw coal enters along the shoot marked " coal and dirt " and falls on to a table, whence it is carried by a stream of water from the adjacent " water box," supplied by the " primary feed pipe," to a second table, and thence to the washing trough proper. The heavier portions of dirt are unable to jump the gap between the two tables and settle into a lower trough, where they are collected by a refuse

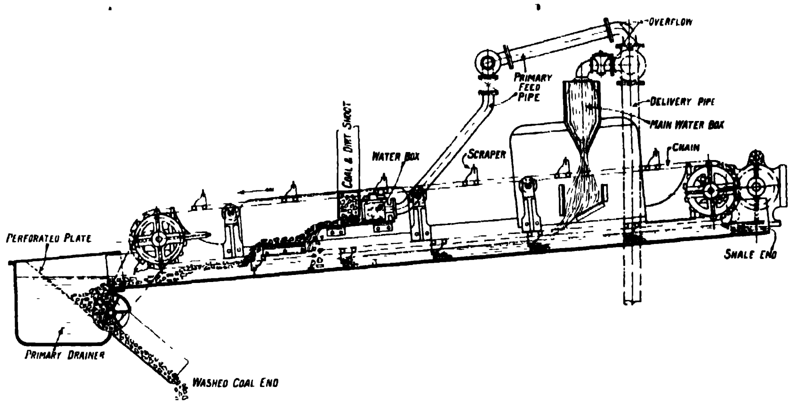

FIG. 85.—Elliott Primary Washer.

scraper. When using this primary washer, the amount of dirt mixed with the coal when it is fed into the washing trough is much reduced and, moreover, the raw coal is thoroughly wetted. With some coals, the introduction of this device enabled the length of the washing troughs to be reduced by as much as 15 ft.

A sectional elevation of an improved Elliott trough washery is given in Fig. 86. Provision is made for the introduction of coal directly from the wagons into the elevator pit, or by means of conveyors, which are loaded by an elevator from an external bunker. The coal is elevated to the top of the washery and passes over a fixed sieve which divides it into two or three sizes, each of these sizes being delivered to a primary washing table where the coarser dirt particles are removed. The thoroughly-wetted and partially-purified coal then passes to a series of troughs where it is washed as previously described. The washed coal passes on to screen conveyors where the water is removed, and the dewatered washed coal then passes to storage. The water passes along the " return water

trough " and enters a settling tank in the base of the washery. Here the finer particles of slurry settle and are collected in a concentrated form by means of a scraper conveyor working slowly along the bottom of the tank. The slurry (or " schlamm ") is elevated by the conveyor, and loaded into the " schlamm " shoot for mixing with the washed coal, or for use elsewhere. The clear water from the

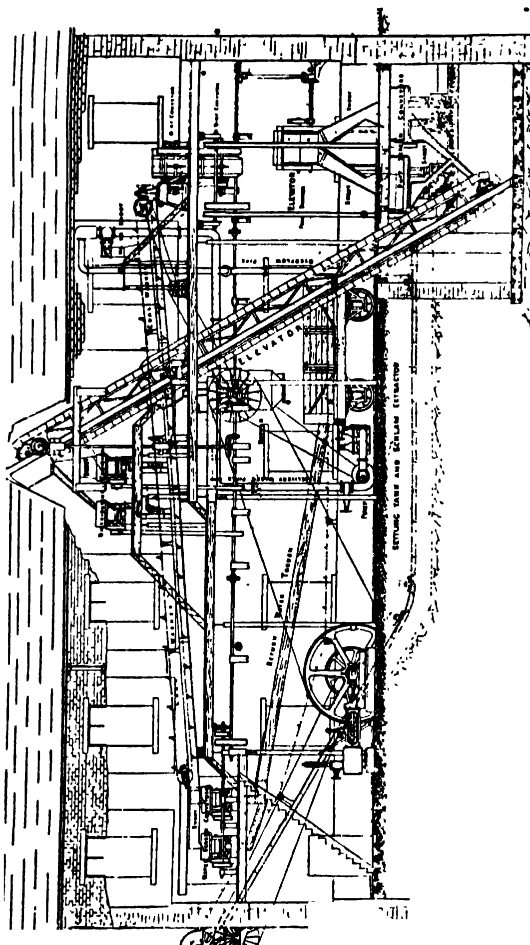

FIG. 86.—Elliott Washery.

top of the settling tank overflows into a pump basin and is pumped back to the troughs for re-use. The dirt passes to the top of the washing troughs on to a conveyor for conveyance to bunkers.

In the improved Elliott washer the troughs may be 27 in. wide at the bottom and 39 in. wide at the top, a double-chain scraper being used for this size of trough, which has a capacity of 10 to 12 tons per hour. A single-chain scraper is used where the width of the trough is 18 in. at the bottom and 30 in. at the top ; such a

trough has a capacity of 6 to 8 tons per hour. The scrapers are spaced 3, 4 or 6 ft. apart and have a depth of 2 to 3 in., the rate of travel being from 14 to 22 ft. per minute according to the size of coal washed. Previous sizing of the coal before washing is an accepted practice with this washer, the sizes adopted being generally 2 to $1\frac{1}{4}$ in., $1\frac{1}{4}$ to $\frac{5}{8}$ in., $\frac{5}{8}$ to $\frac{}{16}$ in., and $\frac{5}{16}$ to $\frac{1}{16}$ in. The dust less than $\frac{1}{16}$ in. is extracted and is not washed. In this practice, the rate of the water current and the speed of the scrapers may be adjusted to suit the size of the coal washed. The Elliott trough washer has therefore been made automatic in action, and a more efficient washing may be effected with its use than with the simple troughs previously described.

Elliott trough washers have been erected to deal with as much as 100 tons of coal per hour in a series of troughs, the power required for this output being 90 h.p. The power requirements for single troughs is said to be 2 h.p. for the smaller trough and $2\frac{1}{2}$ to 3 h.p. for the larger trough. The water requirements are high, amounting to 250 to 320 gallons per minute, or 1,600 to 2,000 gallons per ton of coal. The water is re-used after settling the slurry.

At Wirral, Cheshire (*Trans.* Inst. Min. Eng., 1896, 11, 55), an Elliott washer, of the type first described, washed 10 to 11 tons of

TABLE 68.—WASHING RESULTS WITH ELLIOTT TROUGH WASHERS (1894–95)

Place.	Size of Coal. in.	Ash per cent. Unwashed Coal.	Ash per cent. Washed Coal.	Ash per cent. Refuse.
Ligny-les-Aire, France	$\frac{3}{16}$ to $\frac{3}{8}$	—	4·5	55·7
	$\frac{3}{8}$,, $\frac{5}{8}$	—	4·9	60·0
	$\frac{5}{8}$,, 1	—	3·9	58·4
Bethune, France	0 to $\frac{3}{8}$	—	4·1	45·7
	$\frac{3}{8}$,, $\frac{5}{8}$	—	3·3	60·3
	$\frac{5}{8}$,, $1\frac{1}{4}$	—	1·5	71·2
Rothschild-Gutmann, Austria.	$1\frac{1}{4}$	12·4	6·1	63·7
Liége, Belgium		—	6·6	56·7
	0 to $\frac{5}{16}$	—	7·5	71·1
		—	5·6	57·0
	$\frac{5}{16}$,, $1\frac{1}{4}$		7·1	68·7
	$1\frac{1}{4}$,, $2\frac{1}{2}$	—	4·3	61·6
		—	3·9	62·7
	0 ,, $2\frac{1}{2}$	—	6·2	67·4

coal, of size less than 1 in., per hour. The ash in the coal was reduced from 25 to 4·20 per cent., in one example, the rejected refuse containing 68·75 per cent. of ash. The cost of washing was stated to be 0·875*d.* per ton. Another Elliott washer at New Consolidated Charlotte Pit, Silesia (*Oest. Zeit. für Berg-, Hütten-, u. Salinenwesen*, 1897, 45; 233), had a capacity of about 6 tons of coal per hour, the coal being less than 0·275 in. size. The ash content was reduced from 15 per cent. to 5 or 6 per cent., whilst the dirt removed contained 50 to 60 per cent. of ash. The water consumption was 175 gallons per minute, or 1,750 gallons per ton of coal washed.

Further washing results with the early form of Elliott trough washer are recorded in Table 68.

Comparison of the percentage of ash in the various sizes of dirt for each type of coal shows that (although the cleanness of the washed coal is satisfactory) the chief difficulty of washing coal in troughs, that is, loss of coal in the refuse, was not overcome in the earlier form of Elliott trough washer. Since this time (1894–95) considerable improvements have been made in the design of the Elliott washer, but results are, unfortunately, not available to show the effect of these.

Since 1899, nine washeries using the improved type of Elliott trough washer have been erected, with a total hourly capacity of 331 tons of coal, or an average hourly capacity of 36·8 tons.

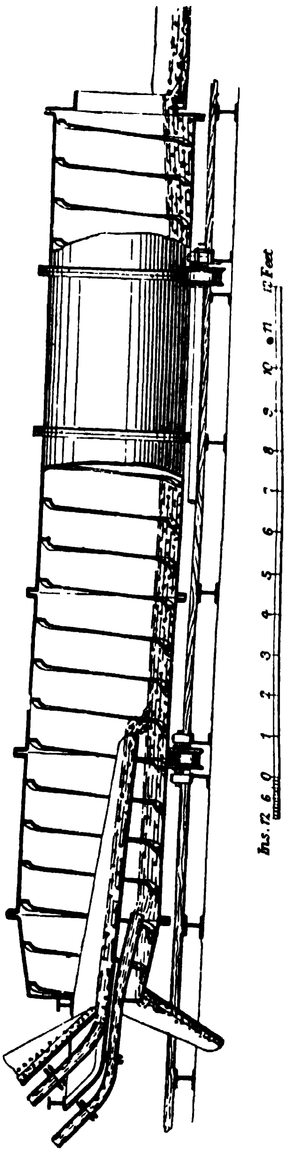

Fig. 87.—The Blackett Washer.

The Blackett Washer.—One of the most ingenious of mechanically-operated trough washers is the Blackett washer, patented by Blackett and Palmer in 1895. It is illustrated diagrammatically in Fig. 87. Fig. 88 is a view of a Blackett washery using six units.

The Blackett washer consists essentially of a cylindrical steel drum, or barrel, fitted with an internal worm or spiral which has a pitch of 18 in. and a depth of 2 to 3 in. The barrel is inclined at an angle equivalent to a slope of 1 in $8\frac{1}{2}$ to 1 in 12, and is supported on

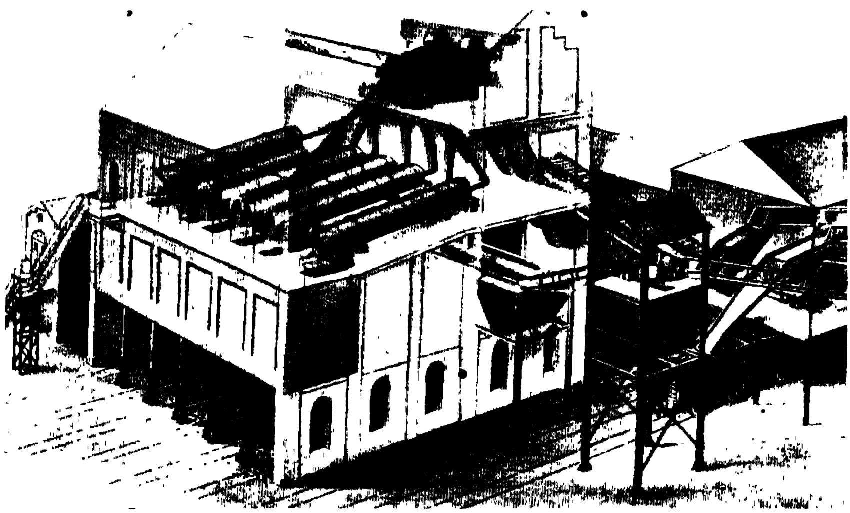

FIG. 88 Stereoscopic View of Blackett Washer

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rollers. The barrel is rotated by a friction drive from a pair of rollers at a speed of 10 to 12 r.p.m.

Coal is admitted at a point 8 ft. from the upper end by means of a shoot, and a supply of water carries the coal into the barrel. A further supply of water is admitted at the upper end of the barrel. The lighter particles of coal are carried by the stream of water over the projecting portions of the worm, which act as dams and arrest the heavier dirt particles. The washed coal is carried from the lower end of the barrel to a drainage screen. The dirt arrested by the worm is carried up above the water stream by the rotation of the barrel until the influence of gravity causes it to slide down again into the stream. The travel of the worm guides the dirt towards the upper end of the barrel, and in time it is discharged into a dirt shoot. The constant sliding along the projecting worm agitates the settled dirt and enables a recovery of most of the coal carried with it.

At Houghton Main Colliery, Yorkshire, considerable attention has been paid to the working of a Blackett washer, and the design has been greatly simplified. We are indebted to Mr. J. Brass for permission to publish some of the results of this work. It has been found possible to reduce the length of the barrel of the slack washer from 28 ft. 6 in. to 13 ft. 6 in., whilst retaining the original diameter of 4 ft. The inclination has been reduced to 1 in 17. The pitch of the spiral is the same as in the original washer, and the depth has been made 3 in. at the lower (washed-coal delivery) end, tapering to 2 in. at the upper (dirt-delivery) end. The capacity of one unit has been raised from 12–15 tons per hour to approximately 18 tons per hour, so that the five units used wash a total of 90 tons of slack per hour.

At Houghton Main, a portion of the scroll (spiral) has been removed from the fines washer where the coal drops from the feed shoot. As a result the bed is not disturbed by the removal of dirt by the spiral in this zone, and the material moves gently up and down the barrel and is stratified by the turbulent motion of the water. The lower layer of dirt is then pushed towards the upper end of the barrel by the dirt advancing from the lower section, until it is caught by the spiral in the upper section and carried away. This modification allows a preliminary stratification of the coal and dirt particles before they enter the washing section proper. This preliminary stratification allows the capacity of the barrel to be increased.

In washing fine coal (0 to $\frac{5}{8}$ in.) in a Blackett washer, the results recorded in Table 69 were obtained.

When washing coal down to 6·75 per cent. of ash, the amount of free coal in the dirt rarely amounted to more than 2 per cent.

The fines washers deal with 90 tons of coal per hour, and require 50 h.p., with a further 30 h.p. for the water pump. The water requirements are 600 gallons per minute or 2,000 gallons per ton of coal.

TABLE 69.—WASHING RESULTS FOR SMALL COAL IN A BLACKETT WASHER

	Unwashed Coal. per cent.	Washed Coal. per cent.	Refuse. per cent.
Ash	15·60	6·75	83·50
Sulphur	1·53	1·30	—
NaCl.	0·598	0·150	—

Each Blackett washer for nut-size coal ($\frac{5}{8}$ to $1\frac{1}{2}$ in.) at Houghton Main Colliery has a length of 18 ft., a diameter of 4 ft., with an inclination of about 1 in 10. The spiral has a pitch of 18 in. and a uniform depth of 2 in. Each barrel rotates at a speed of 12 r.p.m. and has a capacity of 15 tons per hour. The four units used have, therefore, a total capacity of 60 tons per hour, and require 45 h.p., with a further 35 h.p. for the water pump. The water circulation is 800 gallons per minute or 3,200 gallons per ton of coal.

With these washers the ash content of the nuts is reduced from about 12 per cent. to 2·7 per cent. in the size $\frac{5}{8}$ to $\frac{7}{8}$ in., and 4·7 per cent. in the $\frac{7}{8}$ to $1\frac{1}{2}$ in. size, the loss of coal in the dirt being of a similar order to that in the fines washer.

It may be noted that the water requirements are abnormally high, but this is partly on account of the large percentage of salt in the raw coal. To reduce sufficiently the salt content of the coal, which is required for by-product coke manufacture, it is necessary to run to waste some 300 gallons of water per minute from the fines washer. These conditions are abnormal and due regard should be paid to them in considering the results recorded.

According to experience at Houghton Main Colliery, where the total capacity of the washery is 160 to 170 tons per hour, the successful operation of a Blackett washer depends upon the maintenance of a regular and correctly-proportioned supply of water, and an inclination of the barrel determined by experiment.

L. F. H. Booth (*Trans.* Inst. Min. Eng., 1926–27, 73, 107) records the results of washing nut coal at Deaf Hill Colliery, Co. Durham. Six Blackett washers, 30 ft. long, each treat a total of 6 tons per hour, this total being governed to a certain extent by the proportion of large and small nuts present in the raw coal. The coal cleaned is from the Hutton, Harvey and Busty seams, the last-named seam being much dirtier than the others, and is treated separately, the other two seams being mixed together. Fig. 89 is a flow sheet showing the lay-out of the plant, and Fig. 88 a stereoscopic photograph of the same plant. The tubs of coal from the shaft gravitate to a creeper, which discharges them over a weighbridge, and thence to the tipplers. One tippler is for shale (" stone "), which is delivered on to a conveyor running outside and parallel to the

screening plant, and thence to a hopper. The two coal tipplers deliver on to shoots connected with two primary jigging screens, which deliver the coal over 3 in. size on to two picking belts, where the large dirt is removed. The through 3 in. coal is also passed in a separate stream on to the picking belts, which are used as conveyors for this size. The dirt removed by hand-picking passes down shoots which deliver through a cross conveyor to the shale ("stone") conveyor. The hand-picked (over 3 in.) coal is then removed by further jigging screens directly into wagons. The under-size from

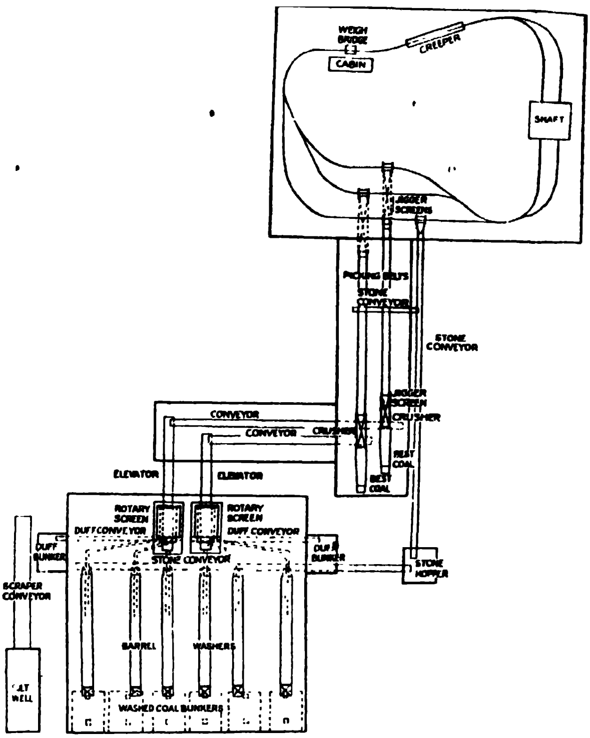


FIG. 89.—Plan of Blackett Washery.

the two screens passes to two cross conveyors, and thence by two elevators to two rotary screens. The two streams of coal are graded into four sizes, of which the smallest, the duff, is by-passed and sold separately. The graded trebles, nuts and peas, for each of the two streams of coal, pass to separate washers.

The inclination of the barrels is 1 in 10·7 for the trebles, 1 in 9·6 for the doubles, and 1 in 8·5 for the singles. All the barrels are rotated at 11 r.p.m. Dog clutches are fitted to each barrel to allow any one to be stopped or started independently of the rest. A length of 4 ft. 9 in. at the delivery end of each barrel is made of perforated plate of $\frac{1}{4}$ in. mesh, with an internal spiral to guide the

nut coal to the hoppers at the end, whilst allowing fines, produced by breakage, to pass with the water through the perforations. The fines so produced pass to a silt well and are removed by a chain-driven scraper-conveyor into wagons. The results of washing are recorded in Table 70.

TABLE 70.—WASHING RESULTS, BLACKETT WASHERY AT DEAF HILL COLLIERY

Seams.	Description.	Size.	Capacity, Clean Coal per hr.	Per cent. Ash in Coal.		Per cent. Coal * in Refuse.	
		in.	tons.	Before Washing.	After Washing.	Free Coal.	After Crushing through ½ in. Mesh.
Harvey and Hutton	Trebles	2 to 3	6	2·68	1·45	Nil.	2·75
	Nuts .	1 ,, 2	12½	10·82	2·28	Nil	1·50
	Peas .	—	9½	—	—	..	-
Busty	Trebles	2 to 3	14	30·07	6·72	Nil	2·24
	Nuts .	1 ,, 2	10	12·02	6·63	Nil	2·55
	Peas .	—	11	—	—	..	.

* Floats at 1·40 S.G.

The figures for peas are, unfortunately, not given. It is not often that the cost of washing would be faced for a coal containing only 2·7 per cent. of ash in the raw coal, although 1·2 per cent. of dirt is removed. The cleaning of the Harvey and Hutton nuts is good, but insufficient evidence is given to judge the efficiency of cleaning the Busty coal. The loss of free coal is *nil*—a surprising result; it would have been interesting to have figures for the floatings in S.G. 1·6 for the refuse in all cases.

The power used to drive the six barrels, for a total capacity of 66 tons of coal per hour, was 50 h.p. The water pumps absorbed 85 h.p. The total power consumption, including 25 h.p. for the feed elevator, was therefore 160 h.p. or 2·6 h.p. per ton. The power required for the whole surface plant was :—

	h.p.
Tipplers and jigging screens	36
Dirt conveyors	25
Coal conveyors and jigging screens . . .	60
Cross conveyors	25
Elevators	25
Barrels	50
Pumps	85
Aerial ropeway	20

The total power was used for a daily quantity of 1,235 tons in seventeen hours. The "bests" coal, over 3 in., amounted to 328 tons, leaving 907 tons, from which 295 tons of duff, through ½ in., was removed. The 3 to ½ in. coal then amounted to 612 tons or 36 tons per hour. The products produced were :—

	Tons.
Washed trebles	90
Washed nuts	218
Washed peas	185
Silt	31
Dirt (14·37 per cent.)	88
Total	612

The costs of producing these fractions, including the "bests" and duff, were :—

	Pence per ton.
Power	2·34
Wages	4·63
All other charges	3·30
Total	10·27

The labour charges were for nineteen men per shift for the screening plant, washery and auxiliary plant. Nine men worked in the washery.

The Murton Washer.—Another ingenious mechanically-operated trough washer was the Murton, invented by Wood and Burnett, and erected at Murton Colliery, Co. Durham (W. O. Wood, *Trans.* Inst. Min. Eng., 1894–95, 9, 42). Whilst in other mechanically-operated trough washers attempts were made to stir the settled refuse, and to obtain continuous working by carrying the settled refuse to the upper end of the trough by travelling scrapers (as in the Longrigg and Elliott washers), or by the action of a worm in a rotating cylinder (as in the Blackett washer), the Murton continuous washer embodied a novel idea in moving the trough itself. The trough consisted of an endless articulated steel belt * composed of "trays," each 3 ft. long, jointed together and made watertight by a suitable arrangement of the joints. The trough was made to revolve slowly round drum heads at the upper and lower ends of the washing section, as shown in Fig. 90. The trough was, in effect, 60 ft. long, the width was 3 ft., and the sides were 8 in. high, the inclination being 1 in 18. Dams 2 in. high were

* Compare with Brunton's belt, introduced over 50 years previously.

fixed at intervals of 3 ft., so that each dam was a part of a joint between two "trays." The trough moved upwards at a rate of 8 to 10 ft. per minute. Coal was admitted to the trough in a stream

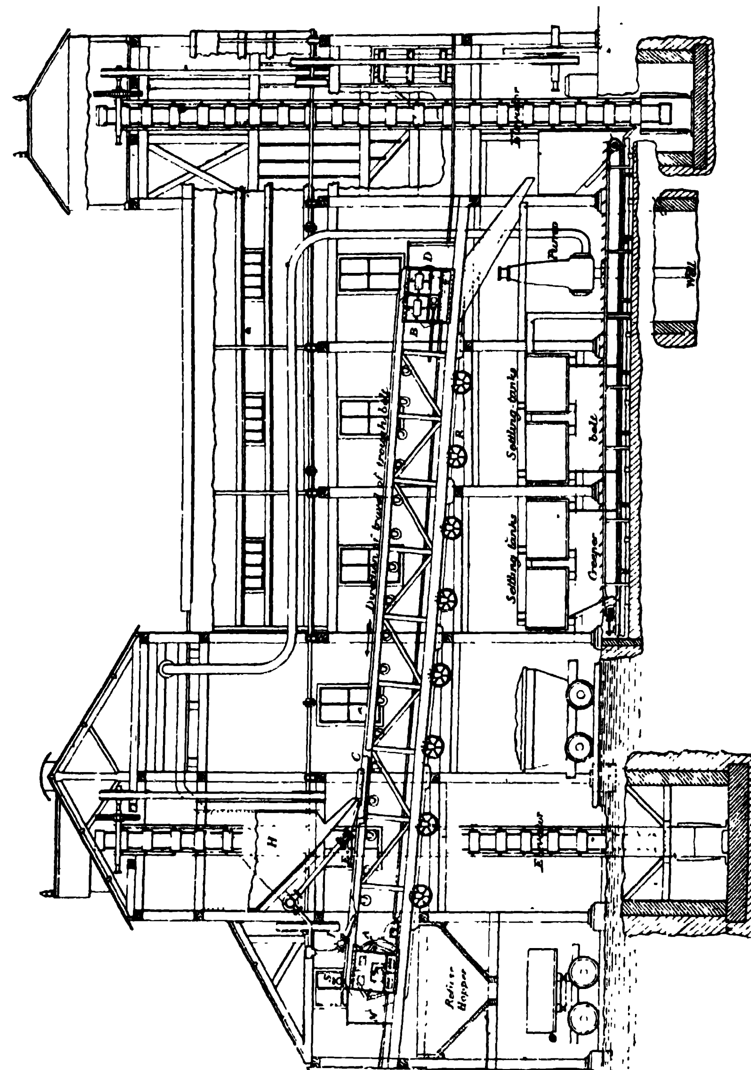


FIG. 90.—Murton Washery.

of water at a point about 15 ft. from the upper end. A second water supply was admitted at the extreme upper end. The washed coal was carried over the lower end of the washing section on to a drainage sieve with $\frac{1}{16}$ in. perforations, over which it passed into the boot of an elevator. The washing water was run into one of

four settling tanks, and the water was re-used. The slurry was removed from the settling tanks by means of a creeper. The refuse settled in the water stream and was arrested by the travelling dams and carried to the upper end of the trough above the point of admission of the raw coal. Here it was re-washed before passing into a refuse hopper.

In the washery at Murton 40 tons of coal were washed per hour in two troughs, the coal being less than $\frac{3}{8}$ in. size. The washed coal was said to contain 3·71 per cent. of ash in one example, but the unwashed coal only contained about 9·3 per cent. of ash. In another sample, the raw coal contained 16 to 17 per cent. of dirt and the amount of coal lost in the refuse was said to be not more than 1 per cent. The power required for the two troughs at Murton was only $1\frac{1}{2}$ h.p. The water used was 450 gallons per minute or 625 gallons per ton of coal washed. A Murton coal washer was in operation also at Glyncastle, South Wales, in 1896, and another example at Nunnery Colliery, Sheffield, dealt with 22 tons of slack per hour.

The earlier forms of trough washer had the advantage of simplicity, an advantage, however, which may only be real when the quality of the labour available is poor. In such an event, however, the capital difficulty of trough washing, namely, the loss of coal with the refuse, is likely to be emphasised. In modern times the loss of such large quantities of coal would alone render the use of a simple trough washer impossible. Moreover, the throughput of a simple trough (say, 10 tons of coal per hour) would be insufficient for all processes except for beehive coke manufacture, which is still carried out on a limited scale in districts where the higher value obtainable for beehive coke compensates for the loss of the by-products and the low output obtained per oven per day. For such plants the low capacity of ordinary trough washers is no real disadvantage, but the probable high losses of coal in the refuse would in many cases make it commercially practicable to adopt one of the more efficient types of trough washer, for example, the Elliott or the Blackett.

In many other processes where only a low throughput of coal is required, the adoption of one of the more efficient types of trough washers might prove to be more economical than the installation of a jig washer. In small instllaations, where the cost of labour is a very important factor, the automatic discharge of the washed coal and the refuse from the Elliott and Blackett washers reduces the amount of attention necessary, whilst, in this respect, minimising the effect of the human factor on the uniformity of the washing results. In an up-to-date plant where an Elliott or Blackett washer is installed, the water may be re-used after settling in suitable tanks, so that the water requirements need not necessarily compare unfavourably with those of jig washers, as they did in older forms of simple trough washers. Moreover, the small power requirements

render the cost of washing low, and the simplicity of construction, the low initial cost and the small number of working parts help to make the cost of repairs a small item. The advantage of other types of washer, such as the Baum and Rheolaveur, is probably more appreciable in large installations washing more than, say, 50 tons of coal per hour, though the Elliott and Blackett washers can be used for large outputs.

CHAPTER XIV

THE RHEOLAVEUR WASHER

General Principles

THE elementary theory of the Rheolaveur washer has been discussed by the late Lambert Lecocq (*Fuel*, 1923, 2, 255). From a consideration of the force applied to a particle by a current of water flowing over a horizontal plane, and the frictional resistance between coal and shale particles and the plane, he deduced a formula for the calculation of the ratio of the sizes of particles which can just be separated. The formula may be expressed :—

$$\frac{r_1}{r_2} = \frac{\mu_2(s_2 - 1)}{\mu_1(s_1 - 1)}$$

using our standard notation. Taking the coefficient of friction for coal (μ_1) as 0·4 and for shale (μ_2) as 0·6, he concluded that the ratio $r_1/r_2 = 6{\cdot}5$.

This elementary treatment is incomplete, as it takes no account of the variety of factors that are introduced in practice. In a Rheolaveur washer, the plane is not horizontal but is inclined, the particles do not move from rest but are given an initial velocity in the trough, and the coal particles are seldom, if ever, in contact with the bottom of the trough. A general description of the theory of coal washing in troughs, including the effects of irregularities of the water current, of the shape and size of the particles and the influence of friction was given in Chapter XII, and it is sufficient here to describe the differences in principle between a Rheolaveur washer and an ordinary trough washer. The differences are twofold ; the first difference is in the method of arranging the particles in the form of a stratified bed, and the second in the method of removing the dirt.

In a simple trough washer, the coal and dirt particles are gradually stratified in the same way that the ebbing and flowing tide segregates the particles on a pebbly beach. The particles do not move very rapidly relative to the sides and floor of the trough, but there is a tendency for some of them to lag behind and to sink to the bottom of the bed, whilst others are accelerated and become concentrated in the upper layers.

In a Rheolaveur washer, the particles are fed into an initial steeply-inclined portion of the trough, in which all of them soon acquire a relatively high velocity. Whilst they are moving with relatively high velocities, they are rapidly able to stratify, and the

stratification is more distinct than in the majority of simple trough washers. There are two reasons for this. Firstly the reaction between the particles and the surface of the trough is reduced. The reaction is measured by $w \sin \alpha$, where w is the weight of a given particle and α is the inclination of the trough. With a high value for α, the reaction is considerably reduced and, because the pressure of the particles on the plane is reduced, the bed is less compact and movement of individual particles in it is easier. Secondly, because of their high velocity, the particles have a considerable kinetic energy. The kinetic energy varies as the square of the velocity, and if the velocity of a given dirt particle is twice that of a corresponding particle in a simple trough washer, its kinetic energy, which enables it to displace a coal particle from the lower layers, is four times as great. The greater energy on a steeply-inclined slope, operating in a bed which offers less resistance to movement, enables a rapid and accurate stratification to be induced.

In operation, it is found to be very advantageous that the length of the steeply-inclined portion of the trough of a Rheolaveur washer be greater than a certain minimum, so that when the particles reach the later and less steeply-inclined portion the stratification is complete. When the stratified mixture reaches the less steeply-inclined portion, and the speed of the water current decreases, the heaviest particles, forming the lowest layer, are deposited. The deposition is brought about in the same way that solid particles in suspension in a river are deposited at bends, or in estuaries where the speed of the river is reduced. Deposition is, however, facilitated, for not only is there less tendency for the current to bear the particles forward, but the reduced inclination causes a greater pressure of the particles on the surface of the trough, and there is therefore a greater frictional resistance to their motion.

In a subsequent portion of the trough the next layer of dirt is deposited, and, the inclination of the trough being further reduced, there is a progressive deposition of particles throughout the length, so that, in succeeding portions of the trough, the deposit consists of large heavy particles, then smaller particles of dirt and large particles of middlings, and subsequently the small particles of middlings.

The difference in the method of producing stratification may therefore be looked upon as being that, in a Rheolaveur washer, the particles are initially more or less in suspension in the current, whereas in a simple trough washer a bed of particles is supported on the plane, and the water current moves among and around the particles. In a simple trough the separation of the layers is brought about by the coal being accelerated, leaving the dirt relatively stationary; in a Rheolaveur, all the particles move rapidly and then, whilst the coal continues to move, the dirt is definitely retarded.

In the majority of trough washers, difficulty is experienced by

reason of the mechanical entanglement of coal particles in the layers of dirt. A particle so trapped has fewer opportunities to be released than with the Rheolaveur system, because the motion of the particles is slower, the bed is more compact, and separation of the layers is attempted before the stratification is complete. In a Rheolaveur the stratification is completed (as nearly as is practicably possible) before any separation of the layers is attempted. The more perfect stratification also enables coals containing relatively large amounts of middlings particles to be washed with ease.

The second essential difference between the Rheolaveur washer and a simple trough washer is the method of removing the layer of dirt from the trough. In place of the mechanical means used in the Elliott, Murton or Blackett washers, the dirt is allowed to fall through apertures in the floor of the trough into special receptacles. It can be understood that, if the dirt fell through an unprotected opening, a considerable quantity of water would also pass out from the trough. The loss of water would not only interfere with the stratification of the particles in the upper layers, but coal particles would also be sucked out by the eddy currents formed at the openings. Water is therefore supplied to the receptacles, some of it passing upwards through the aperture to replace the dirt evacuated from the trough, and the remainder passing downwards with the displaced refuse. The water current passing upwards from the receptacle into the trough not only replaces the dirt removed, but provides an upward current at the aperture which prevents coal particles from falling and being discharged with the refuse. It was this upward current which suggested the distinctive name of Rheolaveur (= current washer) for the refuse receptacles and by which the whole process has been named.

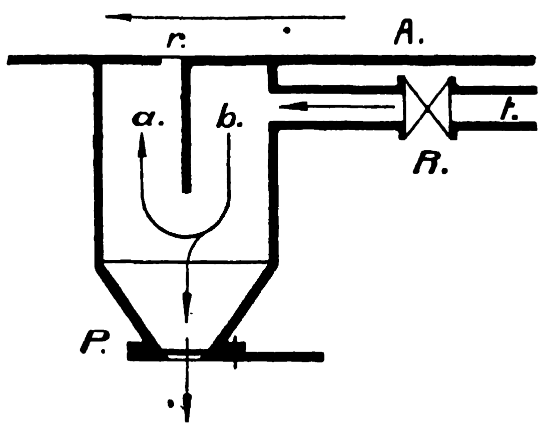


FIG. 91.—Diagram of Rheo-box.

The action of the rheolaveurs, or rheo-boxes, may be understood from Fig. 91. The box consists of a cast-iron chamber in the shape of an inverted pyramid. The uppermost portion forms part of the bottom of the trough, A, and the apex, P, is provided with an aperture for the removal of the refuse. The box is divided by a partition, so that the water from the pipe, *t*, and the cock, R, passes partly downwards through the aperture at P, and partly from the

compartment, *b*, to the compartment, *a*, and through the slot, *r*, into the trough.

This particular form of rheo-box is only used for washing the smaller sizes of coal, less than about $\frac{1}{2}$ in. in size. With larger coal the aperture at P would require to be very large, a 3 or 4-in. aperture being required to discharge 2-in. particles of dirt. With such a large aperture, the amount of water passing away with the refuse would be excessive, and a modified form (sealed type), which will be described later, is employed. The aperture at P is closed by an orifice plate attached to the frame in such a manner that any one of several openings, each with a different diameter, can be used.

There are three standard types of Rheolaveur washer for washing

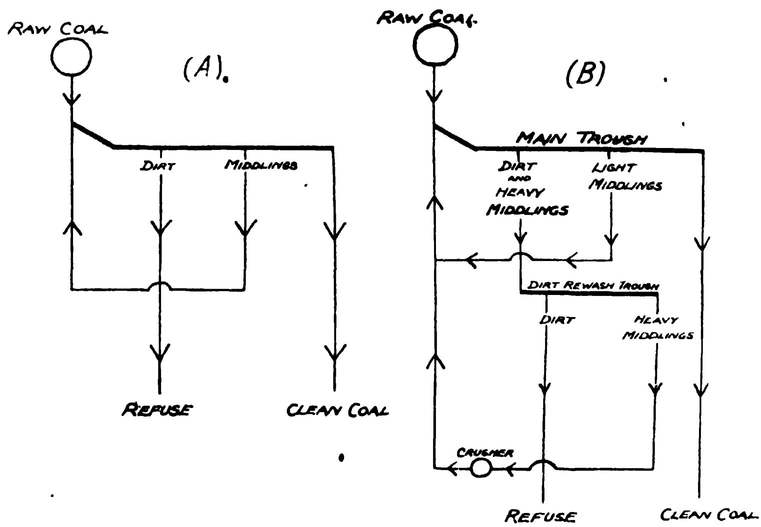


FIG. 92.—Flow Sheets, Rheolaveur Nut-coal Washer.

respectively nuts, fines and slurry, and the Rheolaveur is, at present, the only washer able to clean efficiently any size of coal from 4 in. to 0. Nuts are easier to clean than fines, and fines are easier to clean than slurry. Moreover, a smaller number of troughs and rheo-boxes are required to deal with a given output of large coal than for the same output of small coal.

Before describing the historical development of the Rheolaveur washer, it is desirable to indicate briefly the general scheme of washing adopted in Rheolaveur washers. The earliest units built were fines washers, but an outline of the general scheme adopted in a nut-coal washer will first be given, because the nut-coal washer is simpler and easier to understand.

Frequently, for a capacity of 80 tons per hour of nut coal from 4 in. to $\frac{3}{8}$ in., one trough only is required, and this is fitted with only two rheo-boxes. The shale and heavy middlings are extracted at

the first box, and the remainder of the middlings, together with a small amount of coal, is removed at the second box. The discharge from the second box is returned to the head of the trough for re-washing. With the majority of coals it is usually profitable to recover some of the middlings (up to a specific gravity of 1·6, or even 1·8) with the clean coal, and unless the amount of middlings is unusually small, the discharge from the first box is transferred to a second trough to remove the middlings particles. These recovered middlings are then returned to the main trough, though they are frequently crushed before being so returned to separate the coal included in the interstratified middlings particles.

The flow-sheet in a Rheolaveur nut-coal washer may be represented as in Fig. 92, A being for a coal almost free from middlings, and B for a coal with a large middlings fraction.

According to these flow-sheets all the middlings particles are returned to the main trough, and although a certain quantity of middlings is advertently allowed to collect in the trough, the accumulation does not proceed indefinitely, the lighter passing eventually into the clean coal and the dirtier and heavier into the refuse.

The middlings accumulated in the main trough enable a more accurate stratification of the coal and dirt to be accomplished. With a relatively thick layer of intermediate particles, it is easy for all the heavy dirt to be concentrated in the lower layers, and all the light coal in the upper layers. Although both coal and dirt may be present in the intermediate layer, there will be no dirt particles in the uppermost layers, and no coal particles in the lowest. An intermediate layer of assorted particles is present in all trough washers, as previously explained, but, in the Rheolaveur, a thick layer is intentionally built up to act as a "protective barrier." In the earlier Rheolaveur washers, it was found to be relatively easy to ensure the complete removal of coal from the shale, but until the system of recirculating the middlings was introduced, it was often difficult to ensure an adequate removal of the dirt from the coal. Theoretical grounds for the relative absence of coal from the refuse, namely, the gradual deposition of the lowest layers of dirt, have already been given, and it is because the deposition is only gradual that it is difficult to remove the final few dirt particles from the coal. Towards the end of the trough the bed of material is thinner than at the beginning and, if the trough is "ten particles wide," it is not easy to remove one particle without removing some of the other nine. For this reason some coal is permitted to fall through the final aperture, and is returned to the system, but its amount is minimised by interposing a barrier of middlings between the layer of "pure" coal and the aperture.

In order to understand the lay-out of a Rheolaveur washery, it is essential to appreciate the importance of the system of re-circulating middlings, the system being known as washing in a closed cycle. It

was introduced, in the first place, in order to overcome the difficulty of removing the last particles of dirt from the clean coal, but it has become the most essential feature of the Rheolaveur process because it enables all irregularities, such as a stoppage of the raw coal supply, or a lack of uniformity in the nature or size of the raw coal, to be overcome. In the event of a stoppage, it is only necessary to shut off the refuse-discharge ports and all the coal in the washer is circulated until the feed is restored.

As has been stated, the fines washer is more complicated than the nuts washer. To ensure proper cleaning of the raw coal, three or four troughs must be used, each with a number of rheo-boxes. In the main trough, all the dirt and middlings are removed from the

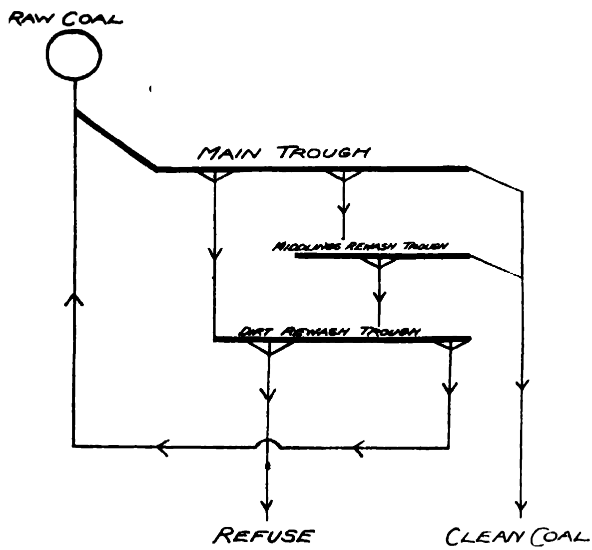


FIG. 93.—Flow Sheet; Rheolaveur Fine Coal Washer.

coal, and clean coal is discharged from the end of the trough. If a very clean product is required, the quantity discharged from the end of the first trough may contain only two-thirds of the raw coal fed to the head of the trough. The material removed through the rheo-boxes in the first trough (which may, in certain circumstances, consist of as much as half of the raw coal), is rewashed in troughs placed vertically below the first. The discharge from the first few rheo-boxes in the main trough is almost entirely composed of dirt, which is still further concentrated by rewashing in the dirt rewash trough. The material recovered, that is to say, the coal and middlings particles passing forward to the end of the dirt rewash trough, is returned to the main trough. The material discharged from the last rheo-boxes in the main trough consists almost entirely of coal and middlings. This is concentrated in a middlings rewash trough, so that only coal, together with any desired proportion of the lighter

middlings, is discharged from the end of this trough. The material evacuated from the rheo-boxes in this rewash trough can be further concentrated in the dirt rewash trough.

A simplified flow-sheet of a Rheolaveur fine-coal washery is shown in Fig. 93. It will be seen that the process is essentially one of gradual concentration. The raw coal is first split into two main fractions, one containing the bulk of the dirt, and the other the bulk of the coal. Each fraction is concentrated so that the refuse contains a minimum amount of useful material, and the clean coal the minimum amount of useless material, and, meanwhile, a third fraction is made consisting of some coal and some dirt, but principally of middlings. This fraction is returned to the main trough, where it helps to build up the "protective barrier," and allows the coal and dirt contained in it a further opportunity for classification.

THE HISTORICAL DEVELOPMENT OF THE RHEOLAVEUR WASHER

A principle somewhat similar to that employed in the Rheolaveur washer was applied in the Bangert washer introduced in 1882. It is illustrated in Fig. 94. In a trough, *a*, there was an opening, *b*, whose

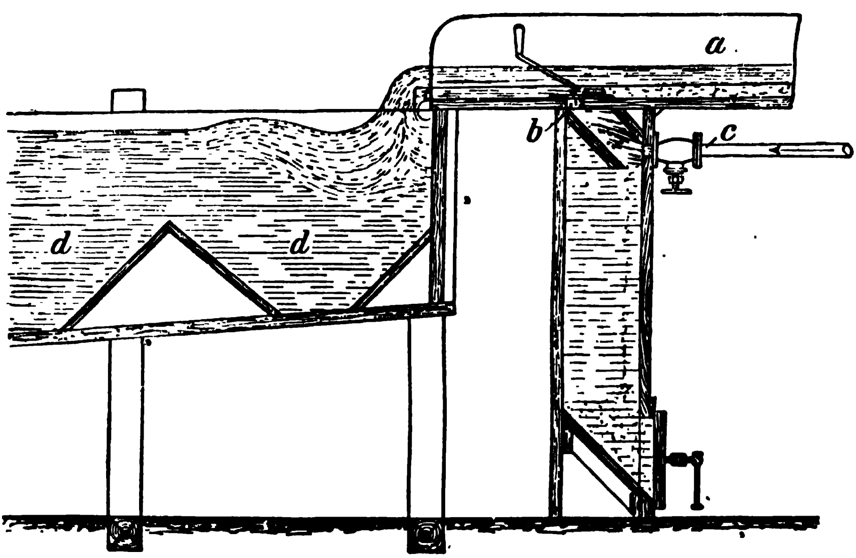


FIG. 94.—The Bangert Washer.

size could be controlled by means of a stopper and handle. The opening was connected with a compartment filled with water from a pipe, *c*. The raw coal entered the trough, *a*, and the lighter coal particles were carried past the opening, *b*, into the spitzkasten, *d*, whilst the heavier dirt particles settled to the bottom of the trough

and passed through the opening, *b*, into the lower compartment. A water current passed upwards through the opening, *b*, to prevent the passage of light coal particles into the dirt compartment. The dirt settled in the compartment and was removed through a bottom door. The partly-washed coal settled in the spitzkasten, *d*, for further cleaning in a jig washer.

The Bangert classifier was used in the Aix-la-Chapelle district, but its possibilities were not fully appreciated, so that it only served as a preliminary cleaning unit for a jig washer. It remained for a Belgian engineer—Antoine France—some thirty years later, to develop the idea of using openings in the bottom of a trough to remove dirt continuously, whilst employing an upward current of water to preserve the stratification and prevent loss of coal with the dirt.

This development took place at the St. Nicholas pit of the Société des Charbonnages de l'Espérance et Bonne Fortune, near Liége, Belgium, a year or two before the Great War began. At this pit the coal contained a large proportion of intergrown coal and shale with ash contents varying from 15 to 60 per cent. This made the coal difficult to wash. The results of washing this coal in Coppée jigs with a capacity of about 60 tons per hour, was, however, considered to be satisfactory until the introduction of coal cutters in the pits enabled a greater coal output to be obtained, and dirtier seams were exploited. The increased coal output could not be washed efficiently with the existing jig washers. The regulation and control of the jig plungers became increasingly troublesome, and a considerable quantity of coal was lost in the refuse. With a view to avoiding the cost of larger buildings and additional coal-washing plant, some experiments were carried out with a trough washer (*strom-apparat*) which was available at the colliery. From these experiments the Rheolaveur washer was developed, and proved to be so successful that it was made the sole means of coal washing.

TABLE 71.—COMPOSITION OF COAL AT ST NICHOLAS PIT (BELGIUM)

Size.		Per cent. by Weight.	Ash per cent.
Mm.	In.		
0 to 0·6	0 to $\frac{1}{40}$	14·5	22·0
0·6 „ 8	$\frac{1}{40}$ „ $\frac{1}{3}$	41·0	22·5
8 „ 20	$\frac{1}{3}$ „ $\frac{3}{4}$	18·2	26·9
20 „ 35	$\frac{3}{4}$ „ $1\frac{1}{3}$	13·6	32·0
35 „ 60	$1\frac{1}{3}$ „ $2\frac{1}{3}$	12·7	31·0

The coal at the St. Nicholas pit (according to Ford, *Trans.* Inst. Min. Eng., 1913–14, 46, 423) had the composition recorded in Table 71. About 30 per cent. of the coal consisted of middlings.

The raw coal, through 60 mm. mesh, was divided into four fractions, namely, 60 to 35 mm., 35 to 20 mm., 20 to 8 mm., and 8 to 0 mm. At first only the two smallest-sized fractions were washed in Rheolaveur washers, the two larger sizes being treated in jig washers. The washed fines were not employed for coking purposes, so that it was found to be most economical to sell the smallest size with an ash content of 10 per cent., and the second size with an ash content of 8 per cent. A middlings fraction was also collected for boiler firing or other purposes, and the refuse was used for hydraulic stowage of the goaf in the mines.

The fine coal of 0 to 8 mm. (0 to $\frac{5}{16}$ in.) size passed from the screens into a trough about 30 ft. long, in which there were eight openings set at intervals of about 1 metre ($3\frac{1}{4}$ ft.), through which to remove the refuse. At each opening, an apparatus (rheo-box) was fitted to allow an upward current of water to be admitted to prevent loss of coal with the dirt which passed through the openings.

The trough of the washer at St. Nicholas pit was 20 in. (500 mm.) wide and 12 in. (300 mm.) deep, with an average inclination of about 1 in 28. The dirt from the first three Rheolaveurs was discarded, and that from the remaining five was rewashed in a second trough of gentler inclination (1 in 115), set below the first. This

TABLE 72.—WASHING RESULTS OF FIRST RHEOLAVEUR WASHER (COAL 0 TO $\frac{5}{16}$ IN.)

	No. of Rheo-box.	Ash per cent. in Product.	Remarks.
In first trough	1	70·3	Final dirt.
	2	60·9	
	3	60·1	
	4	43·1	Middlings for rewashing.
	5	40·9	
	6	33·3	
	7	31·2	
	8	27·6	
In second trough	1	69·0	Final dirt.
	2	67·1	
	3	63·0	
	4	44·2	Final middlings.
	5	31·2	
WASHED COAL FROM—			
First trough		8·5	
Second trough		11·2	

trough was of smaller dimensions, being 10 in. (250 mm.) in depth and width. It was fitted with five rheo-boxes. The first three of these discharged the final dirt and the remaining two the middlings. The washed coal passing from the end of both troughs was mixed and passed to a settling tank, whence it was elevated to a drainage hopper. The composition of the product removed from each rheo-box is recorded in Table 72. The figures recorded are average results of a series of experiments extending over a period of six days.

The second coal fraction, of $\frac{5}{16}$ to $\frac{3}{4}$ in. (8 to 20 mm.) size, was admitted to a trough 10 in. (250 mm.) wide and 14 in. (360 mm.) deep, with an inclination of about 1 in 12. There were only two rheo-boxes fitted to this trough, the first discharging final dirt and the second discharging into a lower trough for rewashing. The lower trough was of similar dimensions and inclination to the upper trough, and was fitted also with two rheo-boxes, the first discharging final dirt, and the second middlings. The washed coal from the two troughs was mixed, drained on a belt conveyor, and loaded into a hopper. The results obtained in a six-days' trial are recorded in Table 73.

TABLE 73.—WASHING RESULTS WITH FIRST RHEOLAVEUR WASHER (COAL $\frac{5}{16}$ TO $\frac{3}{4}$ IN.)

	Number of Rheo-box.	Ash per cent. in Product.	Remarks.
In first trough	1	71·86	Final dirt.
	2	34·90	Middlings for re-washing.
In second trough	1	68·83	Final dirt.
	2	31·87	Final middlings.
Washed coal from—			
First trough		7·3	
Second trough		10·2	

The results obtained with this experimental Rheolaveur washer were an improvement on those obtained previously with jig washers, since the average ash content of the refuse from the two Rheolaveurs was 67 per cent., compared with 60 per cent. in the refuse from the jigs. Moreover, the output had been increased by 15 per cent. and the control of the plant greatly simplified.

The satisfactory results obtained with the new type of washer led to the installation of a second plant for the Société des Mines de Lens, at Pont à Vendin, France, to treat 40 tons of coal of 0 to $\frac{1}{4}$ in. (0 to 6 mm.) size, in one unit, and 40 tons of coal of $\frac{1}{4}$ to $\frac{3}{8}$ in. (6 to 10 mm.) size in another. The general arrangement of the troughs and Rheolaveurs for the larger size of coal is shown in Fig. 95. In

the upper trough there were three rheo-boxes, the first two of which discharged final dirt, and the third discharged a mixed product to a lower trough for rewashing. The lower trough was fitted with two

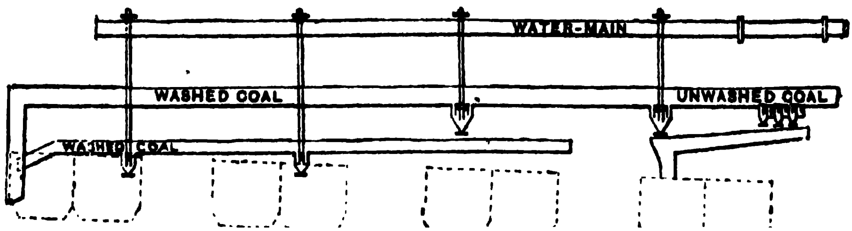


FIG. 95.—Early Form of Rheolaveur Washer (1913).

rheo-boxes, the first to discharge final dirt and the second the final middlings. The washed coal from the two troughs was mixed and conveyed to a hopper. The results of washing are recorded in Table 74.

TABLE 74.—WASHING RESULTS AT PONT À VENDIN (COAL $\frac{1}{4}$ TO $\frac{3}{8}$ IN.)

No. of Rheo-box.	Ash per cent. in Product.	Remarks.
1 (*a*)	80·12	Final dirt.
1 (*b*)	79·28	
1 (*c*)	69·48	
2	74·92	
3	48·30	Middlings for rewashing.
4	70·93	Final dirt.
5	55·33	Final middlings.
Washed coal from—		
First trough	4·70	
Second trough	8·50	

One of the earliest forms of rheo-box used is illustrated in Fig. 96; the plan shows that, in this example, there were four slits or openings in the trough for discharge of dirt. The slits were $1\frac{1}{2}$ in. (40 mm.) wide and extended across the full width of the trough. The four slits communicated with a common discharge orifice at the bottom of the rheo-box.

The orifice was closed by a series of plates, each having an aperture of different diameter. These may be seen in the section through CD. The lower plates, with the smallest apertures, could be swung away to increase the size of the discharge port. In case of blockage whilst in operation, the plate in use could also be swung away until the stoppage was cleared

A current of water from an overhead tank entered the rheo-box at two inlets, shown by dotted lines in the section through EF. One inlet pipe is shown in cross-section in the other cross-sectional

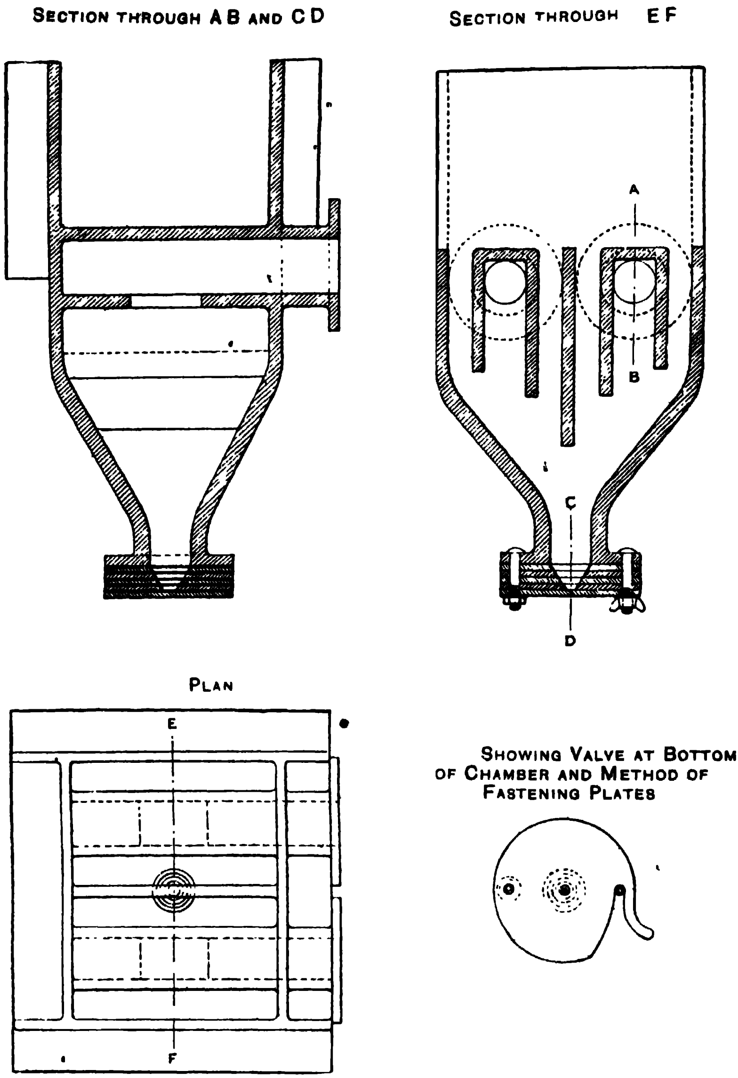


FIG. 96.—Early Form of Rheo-box.

drawing. The water stream entering the rheo-box divided into two portions, as already explained, one passing upwards into the trough to replace the dirt discharged and maintain the stratification, and the remainder passing downwards and carrying the dirt through the discharge orifice.

The dislocation of trade brought about by the Great War—particularly in Belgium—hindered much further development of the Rheolaveur washer until 1918. In that year an acute fuel shortage in France led to a special application of the Rheolaveur washer to treat cinders, and in one year twenty-nine plants were erected for that purpose. These washers were only of small capacity (5 to 20 tons per hour), but the experience gained showed that the Rheolaveur washer was particularly suitable for the treatment of small sizes of inferior material. In 1919, nine Rheolaveur washers were erected, mostly in France, for the treatment of small coal up to 1 in. in size. In 1920, the successful application of the Rheolaveur principle to the washing of large coal led to the construction of washers to treat all sizes up to 2 or 3 in., and in that year thirteen washers were erected—mostly of this type—in France, Belgium, Great Britain and Spain. The first Rheolaveur washer erected in England was at the Ormonde Colliery of the Butterley Company, Nottinghamshire. It was of 100 tons per hour capacity for coal of 0 to 3 in. size. In this year (1920) the Rheolaveur washer may be considered to have firmly established itself as a simple and efficient means for washing large quantities of nut or fine coal, for in the succeeding year (1921), twenty-three washers were erected with an average capacity of 74·6 tons per hour. The development of this washer in the years since the conclusion of the Great War may be judged from Table 75, in which the numbers and capacities of washers built since 1919 are recorded.

In 1924, the Rheolaveur washer was introduced into America, and twelve plants have been erected, chiefly for anthracite washing. The total capacity of these twelve plants is 3,045 tons per hour, or an average of over 250 tons per hour.

TABLE 75.—NUMBER AND CAPACITY OF RHEOLAVEUR WASHERS ERECTED SINCE 1919

Year.	No. of Coal Washers.	Average Capacity, Tons per Hour.	No. of Breeze and Slurry Washers.	Average Capacity, Tons per Hour.	Total Capacity of all Washers, Tons per Hour.
1919	8	35·0	1*	10·0	290
1920	4	75·4	2*	10·0	850
1921	21	81·0	2*	7·5	1,715
1922	22	86·4	—	—	1,900
1923	24	112·9	—	—	2,715
1924	17	92·0	7	11·4	1,550
1925	21	122·2	12	11·3	2,705
1926	11	148·6	19	15·0	1,921
1927	16	150·0	8	9·4	2,275

* Breeze Washers.

The numbers of Rheolaveur washers built in different countries are recorded in Table 76. In this table the total figures since 1913 are given, and the cinder washers (thirty-five) and slurry washers (forty-eight) are recorded separately to give due significance to the average hourly capacity of the coal washers.

TABLE 76.—STATISTICS OF RHEOLAVEUR WASHERS BUILT IN DIFFERENT COUNTRIES SINCE 1913

Country.	Coal Washers.			Slurry and Cinder Washers.		
	No. of Plants.	Total Hourly Capacity (Tons).	Average Hourly Capacity (Tons).	No. of Plants.	Total Hourly Capacity (Tons).	Average Hourly Capacity (Tons).
France . .	52	5,055	97·2	43	340	7·9
Belgium .	47	3,655	77·8	20	281	14·0
U.S.A. . .	12	3,035	252·9	2	25	12·5
Gt. Britain .	17	1,295	76·2	1	15	15·0
Germany .	6	870	145·0	—	—	—
Sarre . .	9	730	81·1	12	113	9·4
Silesia . .	3	725	241·7	—	—	—
Spain . .	7	385	55·0	—	—	—
Other countries	13	1,035	79·6	5	58	11·6
Totals .	166	16,785	101·1	83	832	10·0

In 1924, M. France applied the Rheolaveur washer to clean slurry. So successful was the experiment that over forty slurry washers have been erected on the Continent in the last four years. The first slurry plant in England has recently been erected for the Yorkshire Coking and Chemical Company, Castleford, Yorks., and is giving consistently good results.

In the earliest practice with the Rheolaveur washer the coal was screened before washing, and a number of troughs were used to treat the different sizes of raw coal. In recent times, however, it has been found that better results are obtained if the raw coal is washed without sizing, and the present practice is to wash coal from 4 in. to 0 in a nut-coal washer, the fine coal below $\frac{3}{8}$ in. being usually removed from the washed coal and rewashed in a fines washer.

When the raw coal was sized before washing it was carried to the top of the washery by an elevator and discharged on to jigging screens. The coal was divided into, say, four fractions, each of which was stored in a separate hopper. Typical fractions (at Ormonde Colliery, for example) were as follows: 0 to $\frac{5}{16}$ in.; $\frac{5}{16}$ to 1 in.; 1 to $1\frac{5}{8}$ in.; $1\frac{5}{8}$ to $3\frac{1}{8}$ in. The two smaller sizes were treated

in separate units, and the two larger ones alternately in a third washer.

The modern practice would be to pass all the coal into a single trough with the sealed type of rheo-box. The discharges from the trough would consist of washed coal, in which the sizes below $\frac{3}{8}$ in. would be passed to a separate washer, and refuse, containing all the dirt from $3\frac{1}{2}$ in. to $\frac{3}{8}$ in. and the greater part of the dirt below $\frac{3}{8}$ in. If the washer were of small capacity, it might not be necessary to rewash the smallest sizes of the clean coal, but if the washery had a capacity of 150 tons per hour or more, the small coal would be rewashed, and it would be advantageous also to include a slurry washer.

THE RHEOLAVEUR FINE-COAL WASHER

A particular feature of the Rheolaveur washer is its adaptability. The number of troughs, and the number of rheo-boxes in each trough can be varied to meet differing conditions. The exact arrangement of a Rheolaveur washer depends entirely upon the nature of the coal, principally upon the relative proportions of coal, middlings and dirt, and the distribution of these ingredients in different sizes. Nevertheless, the lay-out is standardised in that,

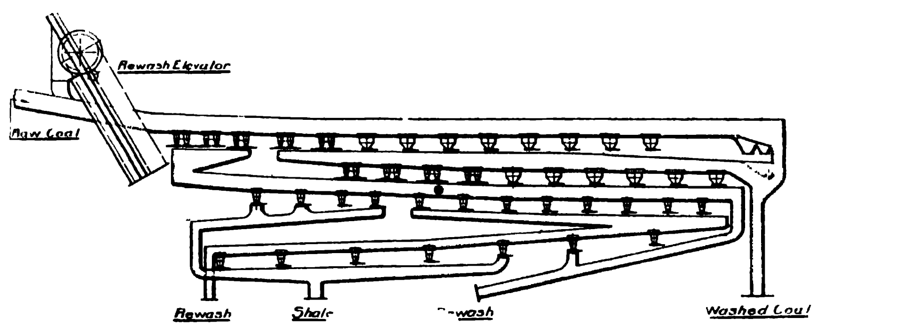


FIG. 97.—" Four-Trough " Rheolaveur Fine-Coal Washer.

for the majority of coals, there are four troughs in the fine-coal washer. This arrangement is shown in Fig. 97.

The course of the various ingredients of the raw coal may be followed in Fig. 97. The upper trough may be referred to as the first trough, and the lower ones as the second, third, and fourth respectively. The function of the first trough is to remove all the dirt and middlings particles so that only clean coal reaches the end of the trough. The second trough receives the bulk of the lighter middlings particles as well as the coal discharged by the rheo-boxes of the upper trough. The products discharged from the ends of the first and second troughs comprise the washed coal. In the third trough the heaviest dirt and heaviest middlings from the first trough, and the lighter middlings from the second trough, are treated, and the

product delivered at the end of the third trough is composed almost entirely of middlings. The object of the fourth trough is essentially the removal of residual middlings particles from the dirt.

The process may therefore be looked upon as divided into two operations. The first operation, in the first and second troughs, is to remove all the useless material and deliver a clean coal of the desired quality. In this operation some of the coal passes through the discharge boxes to the lower troughs. The second operation, in the third and fourth troughs, is to concentrate the dirt. Care is taken that the only material discharged from the system in these two troughs is dirt, free as far as possible from recoverable coal particles. The first operation may therefore be looked upon as a purification of the coal, and the second as a purification of the dirt; any material not discharged, either as clean coal or as refuse, is returned to the first trough by the rewash elevator, where it helps to

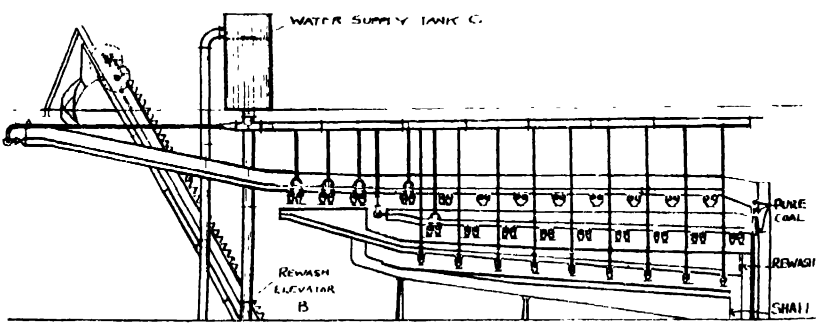


FIG. 98.—"Three-Trough" Rheolaveur Fine-Coal Washer.

form the protective barrier between the "pure" coal and "pure" dirt.

The majority of Rheolaveur fines washers in use, however, are of the older three-trough type. In this older type, the scheme of washing is the same, consisting firstly of the purification of the coal, and secondly of the purification of the dirt The path of the particles is, however, rather different because of the absence of a fourth trough. In a three-trough washer, as illustrated in Fig. 98, the upper trough is about 60 ft. long and is fitted with twelve rheo-boxes, the first four of which only are connected to the overhead water-tank by a pipeline. In the second trough there are nine rheo-boxes, of which only the first is connected to the overhead pipe system. In the third trough, nine rheo-boxes are fitted, all of which are connected to the water-supply system.

In the upper trough, the inclination in the section where the raw coal enters is steeper than in the main length of the trough in order to induce an accurate classification of the raw coal. The first section of the trough is made of rectangular cross-section (Fig. 99) to ensure conditions as uniform as possible over the whole length of

the trough. In the succeeding sections of the trough the inclination is reduced ; most of the heavier dirt particles are already at the bottom of the trough immediately at the end of the initial steeply-inclined portion and, further along the trough, the gentler inclination allows the lighter dirt particles to settle. The cross-section of the main length of the trough is trapezoidal (Fig. 100) in order to concentrate the heavier dirt particles and increase the depth of the bed of heavier particles settling on the bottom whilst still allowing the lighter coal particles above them an ample freedom of movement. With this shape of trough a greater quantity of material can be treated efficiently than in a trough of simple rectangular cross-section. As the rheo-boxes reject a portion of the dirt, the quantity of material to be handled in the succeeding portion of the trough diminishes, and the width of the trough is progressively decreased.

At the end of the first section of the upper trough, the heavier dirt has settled to the bottom and is rejected through rheo-boxes 1, 2 and 3 directly to the third trough (Fig. 98). The classification

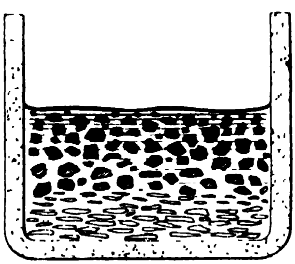


FIG. 99.—Cross-section of First Portion of Upper Trough. Rheolaveur Fine-Coal Washer.

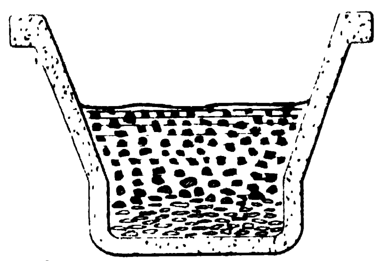


FIG. 100.—Cross-section of Main Portion of Upper Trough. Rheolaveur Fine-Coal Washer.

which is made in the remaining length of the trough is such that lighter dirt particles of decreasing ash content gradually settle to the bottom and are rejected through rheo-boxes 4 to 12. To ensure that no dirt passes away with the clean coal from the end of the first trough, it is arranged that the last two boxes, 11 and 12, reject a certain amount of coal together with all the dirt remaining in the trough. The rejected material from rheo-boxes 4 to 12 passes into the second trough, where it is rewashed. Only the first of the nine rheo-boxes fitted to this second trough is connected to the overhead water tank, and consequently some of the lighter material from the upper layers of the trough is rejected, and only a relatively small amount of material, consisting almost entirely of coal and the lightest middlings, reaches the end of the trough. The remainder, which contains a high proportion of middlings, and frequently also some coal, is rejected, through rheo-boxes 13 to 21, and passes into the third trough. The clean coal which reaches the end of the second

trough joins the material removed from the first trough and passes to the washed-coal sump.

The third trough receives the material rejected from the second trough, and also the heavy particles from the first three boxes (1 to 3) in the first trough. All the rheo-boxes in the third trough are connected to the overhead water tank, and the particles rejected comprise the refuse. In order to prevent rejection of lighter middlings particles with the refuse, it is arranged that a portion of the material from the third trough passes from the end to a "rewash" sump. This material is collected by a "rewash" elevator (or by the raw coal elevator) and passes down a shoot to the upper end of the first trough, and is again subjected to the whole washing process.

The washed coal is collected from the sump by the washed-coal elevator and is elevated to a scraper conveyor for loading into the hoppers. The dirt is also elevated from its sump to hoppers for disposal. The water from the washed coal, rewash and dirt sumps overflows into a series of settling tanks shaped like spitzkasten. The heavier particles of the slurry settle in these settling tanks and are continuously removed through cocks at the bottom, to be rewashed, or mixed directly with the washed coal or the refuse according to circumstances. In a complete Rheolaveur plant they are rewashed in a slurry washer. The clear water from the top of the settling tanks is pumped to the overhead tank for re-use.

The efficiency of washing fine coal in this apparatus may be illustrated by means of analyses we have made of the products from a "three-trough" Rheolaveur washer, which are recorded in Table 77.

TABLE 77.—WASHING RESULTS OF A "THREE-TROUGH" RHEOLAVEUR FINES WASHER

Size (in.)	Unwashed Coal.			Washed Coal.			Per cent. of Dirt Removed
	Per cent. of Size.	Dirt.		Per cent. of Size.	Dirt.		
		Per cent. of Size.	Per cent. on Total.		Per cent. of Size.	Per cent. on Total.	
$\frac{5}{16}$ to $\frac{1}{16}$	50	28·0	14·0	53	0·75	0·4	97·0
$\frac{1}{16}$,, $\frac{1}{45}$	32	34·1	10·9	31	0·93	0·6	95·5
$\frac{1}{45}$,, 0	18	31·1	5·6	14	7·15	1·0	82·0
Total .	100	—	30·5	100	—	2·0	—

The efficiency of separation of dirt from the coal above $\frac{1}{45}$ in. is very good, especially for the sizes above $\frac{1}{16}$ in. The removal of dirt

from the material below $\frac{1}{45}$ in. is also good, and much more satisfactory than can be obtained in the majority of washers unless designed for slurry washing.

Float-and-sink tests were also carried out on a number of " snap " samples of the washed coal and the refuse to test the uniformity of working. The results of these tests are recorded in Table 78.

TABLE 78.—FLOAT-AND-SINK TESTS ON WASHED COAL AND REFUSE

Test No.	1.	2.	3.	4.	5.	6.	Average.	Average (No. 1 Test Excepted).
Time . . .	12·30 p.m.	2.0 p.m.	9.30 a.m.	1.0 p.m.	2.30 p.m.	4·0 p.m.	—	—
Per cent free coal in refuse . .	2·7	3·0	3·0	1·5	4·7	1·7	2·76	2·78
Per cent. free dirt in washed coal.	8·4	0·5	2·6	1·3	1·8	1·8	2·56	1·48

The material recorded as " free coal in the refuse " (floating in S.G. 1·48) always contained a large percentage of dust, which, expressed as a percentage of the refuse, amounted on the average to 1·04 per cent. This dust was accumulated from the washing water. Excluding this coal dust, the average amount of free coal lost in the refuse would be 1·74 per cent. In No. 1 test, the amount of free dirt (sinkings in S.G. 1·48) passing away with the washed coal was abnormal, possibly owing to the temporary blockage of a discharge orifice of one of the rheo-boxes. When this abnormal figure is excluded from the average, the amount of dirt passing away with the coal is very low.

Further tests were made to examine the refuse from the various rheo-boxes of the third trough. In this example of a " three-trough " fines washer there were only six Rheolaveurs in the third trough. and the analyses were made of the combined rejects from succeeding pairs of rheo-boxes. The results of the tests are recorded in Table 79.

TABLE 79.—EXAMINATION OF REFUSE FROM FINAL RHEO-BOXES

		1 and 2	3 and 4	5 and 6
No. of rheo-boxes (final trough) .		1 and 2	3 and 4	5 and 6
Per cent. free coal in refuse . .		1·8	1·3	1·9
Refuse, per cent. of size.	over $\frac{1}{18}$ in. . . .	70	67	51
	$\frac{1}{18}$ to $\frac{1}{45}$ in. . .	22	25	39
	thro' $\frac{1}{45}$ in. . . .	8	8	10

These figures show how the largest size of dirt tends to settle in the first section of the trough, whilst the smaller-sized material settles in succeeding sections.

The "four-trough" fines washer illustrated in Fig. 97 may be considered to be similar to the one described (Fig. 98) so far as the first three troughs are concerned. In the "three-trough" washer the rejected material from the third trough passes directly to waste; in the "four-trough" washer, however, the material rejected from only the first four rheo-boxes of the third trough passes directly to waste, and the reject from the remaining eight rheo-boxes passes from a collecting trough to a fourth washing trough. This is fitted with seven rheo-boxes, the first two of which reject material for rewashing, and the remaining five reject final dirt. The material

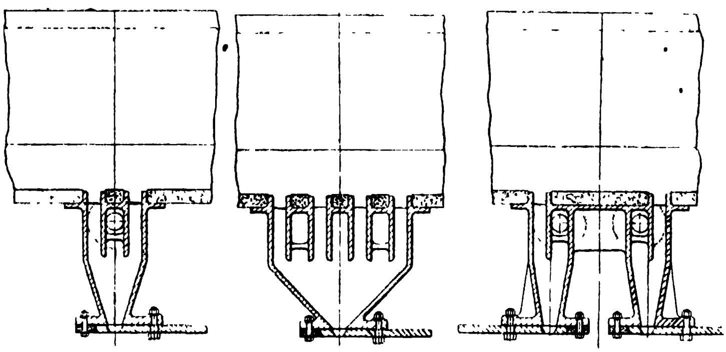


FIG. 101.—Various Forms of Rheo-box Fine-Coal Washer.

which passes from the end of the fourth trough passes to the rewash sump.

The addition of a fourth trough makes the washing process almost mechanical and therefore allows for the variation in control due to the personal element, and ensures a general high efficiency. In many washeries good results are recorded during periods when the washing process is under careful supervision, but in the absence of a close control, the efficiency of washing may be considerably reduced. Thus, although the "three-trough" fines washer, under careful supervision, gives very good results, as already shown, the addition of a fourth trough enables slightly better results to be obtained with less careful control.

The rheo-boxes used in both types of fines washer are, in principle, the same as is shown diagrammatically in Fig. 96. They may, however, be fitted with one or two openings for the discharge of material from the trough. Such boxes are described as being of the single or double discharge type, according to the number of discharge openings provided. Double-discharge rheo-boxes are used in a "four-trough" fines washer at the upper ends of the first two troughs, where the amount of discharged material is the greatest.

Single-discharge rheo-boxes are fitted elsewhere. The different types of rheo-boxes used are illustrated in Fig. 101 in sectional elevation. These different types of rheo-boxes will be readily understood from these diagrams by comparison with Figs. 91 and 96. The orifice plate of the modern rheo-box is fan-shaped with a suitable extension for use as a handle. In the wide portion of the plate there are several holes, any one of which may be placed under the discharge orifice of the rheo-box proper. The holes in each plate have different diameters to control the amount of material discharged by each rheo-box. In the trough, the bed of dirt becomes thinner and the specific gravity of the dirt becomes less the greater is the distance from the head of the trough. At each rheo-box, therefore, a different condition exists and a suitable aperture in the plate can be used at the bottom of each rheo-box to give a discharge appropriate to the conditions at that point. If necessary also, the discharge can be suppressed. If the hole in the orifice plate should be blocked by an aggregation of dirt particles, the blockage may be readily removed by adjusting the orifice plate so that a fresh hole is in communication with the rheo-box.

The troughs are usually made of mild steel, concrete or concrete-lined steel sections. In Belgium, where cheap glazed tiles are available, these are used for the lining of the trough; when using these tiles the inclination of the troughs is reduced because of the lower coefficient of friction of coal and dirt on a glazed tile surface.

Remarkably good results are achieved in the American Rheolaveur fine-coal washer installed in 1926 for the American Smelting and Refining Co., at Cokedale, Colorado. The screen analysis of the raw coal is shown in Table 80, and the results of float-and-sink analysis in Table 81. The washer is of the four-trough type.

TABLE 80.—SCREEN ANALYSIS, COKEDALE COAL

Size.	Weight per cent.	Ash per cent.
> $\frac{5}{16}$ in.	0·2	28·8
$\frac{5}{16}$ in. to $\frac{1}{4}$,,	2·6	16·5
$\frac{1}{4}$,, ,, $\frac{1}{8}$,,	23·2	17·0
$\frac{1}{8}$,, to 14 mesh	30·4	16·5
14 mesh ,, 48 ,,	28·1	16·0
48 ,, ,, 100 ,,	7·7	19·7
100 ,, ,, 200 ,,	3·9	21·5
< 200 mesh	3·9	22·6

This coal is evidently difficult to wash; not only does it contain about 16 per cent. of size less than 48 mesh (Tyler standard, about

TABLE 81.—FLOAT-AND-SINK ANALYSIS, COKEDALE COAL

S.G.	Weight per cent.	Ash per cent.
< 1·32	47·5	6·0
1·32 to 1·36 . . .	25·0	10·2
1·36 „ 1·48 . . .	8·5	20·5
1·48 „ 1·6 . . .	7·5	34·0
> 1·6 . . .	11·5	61·5

$\frac{1}{80}$ in.), but in order to obtain a yield of 70 to 80 per cent. it is necessary to recover material up to 20 per cent. of ash, and the washed coal must contain about 10 per cent. of ash. The washery was erected with a guarantee to yield 78 per cent. of clean coal with an ash content of 10·8 per cent., the refuse containing 48 per cent. of ash. In operation during six months, it gave an average yield of 80 per cent. with an ash content of 10·5, the refuse containing 47·8 per cent. of ash (Walle and Woody, *Min. Cong. Jnl.*, 1927, 13, 197).

One of the most interesting results, however, is the degree of washing of the small sizes. The average ash contents of different sizes of the raw coal, washed coal and refuse in 8 tests are given in Table 82.

TABLE 82.—AVERAGE ASH CONTENTS, COKEDALE COAL

	Total.	Above $\frac{1}{16}$ in.	$\frac{1}{16}$ to $\frac{1}{80}$ in.	Below $\frac{1}{80}$ in.
Raw coal .	18·5	18·2	18·2	19·8
Washed coal .	10·5	9·4	10·6	17·1
Refuse . .	49·1	49·5	48·5	47·4

These results indicate that the washer was able to wash satisfactorily coal from $\frac{5}{16}$ to $\frac{1}{80}$ in. The coal below $\frac{1}{80}$ in. was not appreciably cleaned, but the refuse of this size contained almost as much ash as that from the largest sizes, indicating that, although the coal through $\frac{1}{80}$ in. was only slightly improved in quality, no loss of coal was sustained.

That the washed coal contained 10 per cent. of ash, and the refuse only 49 per cent. of ash is no reflection on the washer, because this is what it was set to do, and it is doubtful if any washer could have improved upon the results.

THE RHEOLAVEUR NUT-COAL WASHER

The Rheolaveur nut-coal washer is of very simple construction, and may, indeed, consist merely of a single trough 30 to 40 ft. in length, fitted with only two rheo-boxes, the first rejecting final dirt and the second the remaining dirt with a proportion of lighter material for rewashing. An elevation of a nut-coal washer is given in Fig. 102. It will be observed that, as in the fines washer, the inclination of the first portion of the washing trough is steeper than the succeeding length. The more steeply-inclined portion may be 16 to 20 ft. long, and serves to give the raw coal an initial velocity and to produce a fairly accurate stratification. The succeeding length of the trough may be only of the same length as the first steeply-inclined section, for the classification of coal and dirt is easier, the larger the size of material treated. One feature of a nut-

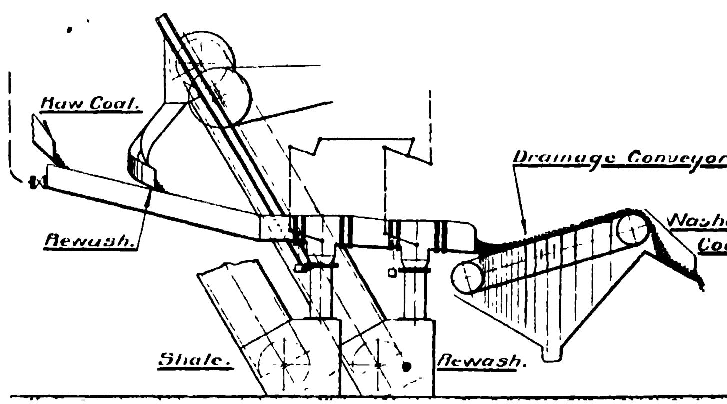


FIG. 102.—Rheolaveur Nut-Coal Washer.

coal washing trough is that it requires about the same ground space as a picking belt, and may indeed be conceived to act as a mechanical picking belt, operating more efficiently than is possible with human labour.

The rheo-boxes employed for nut-coal washing are different in construction from the simple type shown diagrammatically in Fig. 91 and used for fines washers. As has been explained, a discharge orifice large enough to allow free removal of the dirt from nut coal would also allow excessive quantities of water to pass through it. Arrangements are therefore made for an intermittent dirt removal. These arrangements are illustrated in Figs. 103 and 104.

Instead of the "open type" discharge box used in the fines washer, the nut-coal rheo-box is a closed box mechanically operated. The particles discharged pass into the boot of an elevator, as shown in Figs. 102 and 105, and the elevator itself is enclosed and filled with water to about the same level as the trough.

Instead of a slit in the floor of the trough, as in the fines washer, the trough is connected by a larger opening to a chamber, A (Figs. 103 and 104). The chamber, A, is formed by the inclined grate, *b*, to which a flap, *c*, is hinged at, *d*, and by the plate, *e*. The width of the aperture is adjusted by altering the position of *e* by the lever, *f*. The discharge of shale is permitted intermittently by the operation of the flap, *c*, which is alternately opened and closed by a connecting rod from an eccentric. The flap-valve, *c*, is attached to a weighted lever which is actuated by a chain attached to a second lever. This lever is driven by the eccentric. Water is admitted to the underside of the chamber, A, by one or more cocks.

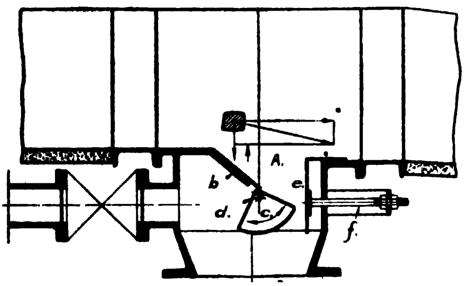


FIG. 103.—Sectional Elevation of Rheo-box used in Nut-Coal Washer.

To control the working of the nut-coal washer, adjustment is

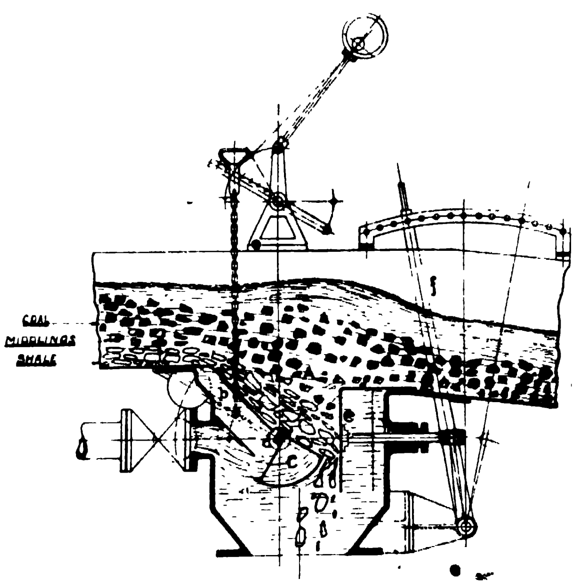


FIG. 104.—Rheo-box in Nut-coal Washer showing the Mechanism for the Intermittent Discharge.

made of the upward water currents and of the operation of the flap valves. By these means loss of coal in the refuse rejected by the first rheo-box is obviated. The only mechanically moving part of the nut-coal washer is the eccentric mechanism to govern the opening and closing of the flap valves.

It has already been stated that, if the raw coal contains much middlings, it is advisable to use a second trough to rewash the shale and middlings. These troughs may be arranged one under the

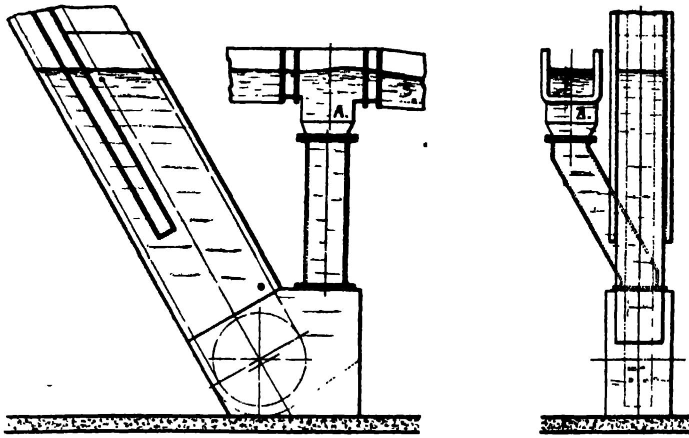
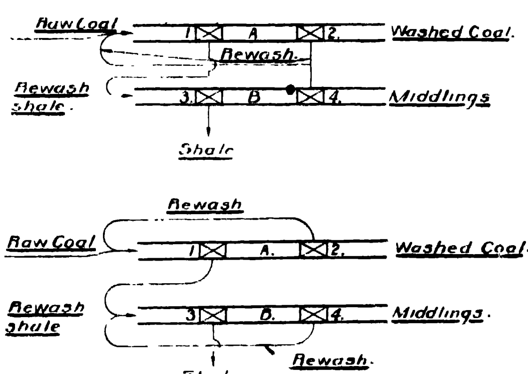


FIG. 105.—Connection between Rheolaveur Trough and Elevator.

other, or side by side, according to the nature of the site available; usually they are placed on the same level. It is possible to arrange the discharge from the rheo-boxes so that either three or four



FIGS. 106 AND 107.—Two methods of combining Rheolaveur Troughs for Rewashing.

elevators are used. One system is illustrated diagrammatically in Fig. 106; in this system the raw coal enters the first trough, A, and a mixture of shale and middlings is removed from the sealed rheolaveur (1) into an elevator which discharges it at the head of trough B. The second sealed rheo-box of trough A, extracts the remaining middlings together with a small quantity of coal to

ensure that only clean coal passes from the end of this trough. By means of an elevator, the mixture extracted from the second rheo-box of trough A, is returned to the beginning of the same trough for rewashing. The products finally discharged from the first trough are washed coal and a mixture of shale and middlings. This mixture is rewashed in trough B. The first Rheolaveur (3) of this trough discharges final shale, and the second Rheolaveur (4) discharges a mixed product for rewashing in the same trough. The products of this trough are therefore final dirt and a middlings product. This method of washing involves the use of four elevators with the washer.

Fig. 107 illustrates diagrammatically another arrangement of two troughs which only involves the use of three elevators. The general arrangement is similar to that shown in Fig. 106 with the exception that the rewash material from both troughs (that is, the

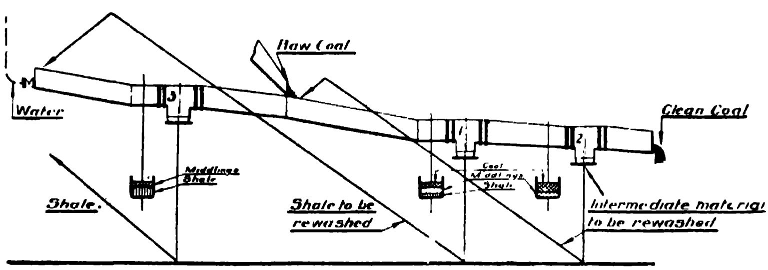


FIG. 108.—Method of Washing and Rewashing in One Trough. Rheolaveur Nut-Coal Washer.

discharged material from rheo-boxes 2 and 4) pass to a common elevator and are rewashed together in trough A.

The method of using two troughs may be considered as effecting, primarily, a separation of the raw coal into two fractions, one containing most of the coal and the other most of the dirt. The subsequent washing of these two fractions is simplified, and the process becomes almost mechanical in operation. For this reason the method is recommended even when dealing with an easily washed coal, for it provides a factor of safety against all the contingencies which may reduce the general efficiency of washing. Moreover, it enables the middlings to be mixed with the washed coal, to be crushed and rewashed, or to be made into a separate product.

An arrangement which fulfils much the same purpose as the two-trough nut-coal washer is illustrated in Fig. 108. In this, the single trough is extended behind the position of entry of the raw coal and the dirt and middlings removed at rheo-box No. 1 are returned to the trough above the point of entry of the raw coal.

This arrangement does not allow of the same amount of regulation as the two-trough washer, and for the majority of purposes the two-trough unit is preferable.

It has been mentioned that the Rheolaveur is adaptable to suit different circumstances, especially in the disposition of the troughs. For this reason it is more suitable for erection in an existing building or at a colliery whose surface plant has already been planned than are many other types of washer, and it is interesting to record that many of the plants on the Continent have been erected to replace existing jig washers. Several conversions have also been made in England. Usually, the Rheolaveur can be fitted into the jig washery building, and in one case, at the Charbonnages de Bray, at Binche, in Belgium, a Rheolaveur washer with a capacity of 200 tons per hour was erected in the building housing a Baum jig washer of smaller hourly capacity. The erection was completed without interrupting the working of the jig washer, and, on changing over, washing was interrupted for only half an hour.

The nut coal washery erected by the Rheolaveur Washery Company for the Barnsley Main Colliery Co., Ltd., serves to illustrate the adaptability of the washer, for none of the standard designs could be housed in the space available.

The design had to be suitable for the raw coal to be delivered to the washer from one end of the picking tables and for the washed coal to be returned to the other end of the picking tables. The refuse had to be discharged at a convenient position for collection with the dirt from hand-picking the lumps, and these requirements limited the length of the building. The inclination of the raw coal conveyor imposed limitations as to height, and the width was strictly limited by the proximity of the London and North-Eastern Railway passenger line.

The coal supplied to the washer was between 3½ and ¾ in. in size and contained about 22 per cent. of middlings of S.G. 1·4 to 1·8. These middlings were collected separately, crushed and rewashed

The lay-out of the plant is shown in Fig. 109. The raw coal from the screening plant is delivered by a belt conveyor to the storage hopper, A, whence it is admitted through an adjustable door, B, into the main washing trough, D. A second door, C, is fitted to the hopper but is not used. The cleanest coal passes along the trough to the lip, E ; the bulk of the water accompanying it drains through the perforated base into the settling tank, F, and the coal passes into the circular double-barrelled screen, G. It is there sized into treble nuts (3½ to 1¾ in.), "double-single" or D.S. nuts (1¾ to ¾ in.), and smalls (below ¾ in.). The two nut fractions can be loaded separately into wagons by the shoots, H, or can be mixed on the conveyor belt, J, and returned to the main screening and loading plant. The coal less than ¾ in. in size falls into the settling tank, F, and is removed by a mild steel worm conveyor, I, to the foot of an elevator, W. It is delivered by the elevator to a jigging screen, Q, which divides it into small nuts (¾ to ⅜ in.) and slack (⅜ in. to 0). The small coal, during screening, is sprayed with water under high pressure to remove the adherent films of dirt and to assist screening.

The washed small nuts and slack are stored in drainage hoppers, X and Y, and loaded directly into wagons.

The shale and heavy middlings removed by the first rheo-box, K, are raised by the elevator, O, to the head of the shale rewashing trough, N. The shale is separated through the rheo-box M. The quantity of water admitted to the base of the rheo-box is regulated by the stop-cock and the box is kept full of shale. Only material of relatively high specific gravity can succeed in working its way through the bed in the box and being discharged as refuse.

The refuse is removed continuously from the base of the rheo-box M, by an endless scraper conveyor, P, which travels along a totally enclosed duct. The floor of the conveyor is inclined upwards on each side away from the base of the rheo-box (as shown in the sectional elevation), to the height necessary to balance the pressure of water applied at the base of the box. The upward inclination of the conveyor is continued to allow the excess of water carried forward in the refuse to flow backwards into the enclosed duct. Further drainage is possible through a perforated section in the floor of the conveyor, near to its uppermost point.

The material passing forward along the shale rewashing trough is delivered to the crusher, S, and falls into the head of the main washing trough, D. The lighter middlings not removed at the first rheo-box, K, in the main trough, are evacuated at the second box, L, and are transferred by the elevator, R, and the shoot, Z, to the head of the main trough for rewashing.

The water-distributing pipes are shown in Fig. 109. The main supply pipe, U, runs horizontally on the top floor of the building and delivers to the shale extracting box, M, and the head of the shale rewashing trough, N. The supply to M is regulated by a cock, and that to N by a valve. A branch pipe, V, leads water from the overhead main to one on the lower floor to supply water to the two rheo-boxes, L and K, and to the head of the main trough, D. The water from the end of the shale rewashing trough drains through perforated plates in the floor of the trough into the main washing trough. The bulk of the water from the main trough drains, at the lip, E, into the settling tank, F. The excess of water from the small coal jigging screen, Q, also drains into the settling tank, and the pump re-circulates the water through the overhead tank, T.

The make-up water is supplied by a small auxiliary pump which delivers water as sprays to the rotating and jigging screens. The capacity of the water-supply tank is 1,120 gallons. In operation, the troughs and water-supply pipes contain 1,300 gallons, so that 2,420 gallons are in circulation in all. The main water pump has a rated duty equivalent to 2,300 gallons per minute against a static head of 46 ft.

The rheo-boxes are all of the single-discharge type (see Fig. 103). Water is admitted to the boxes by one or more cocks. The box, M, in the shale rewashing trough has only one water inlet. It is

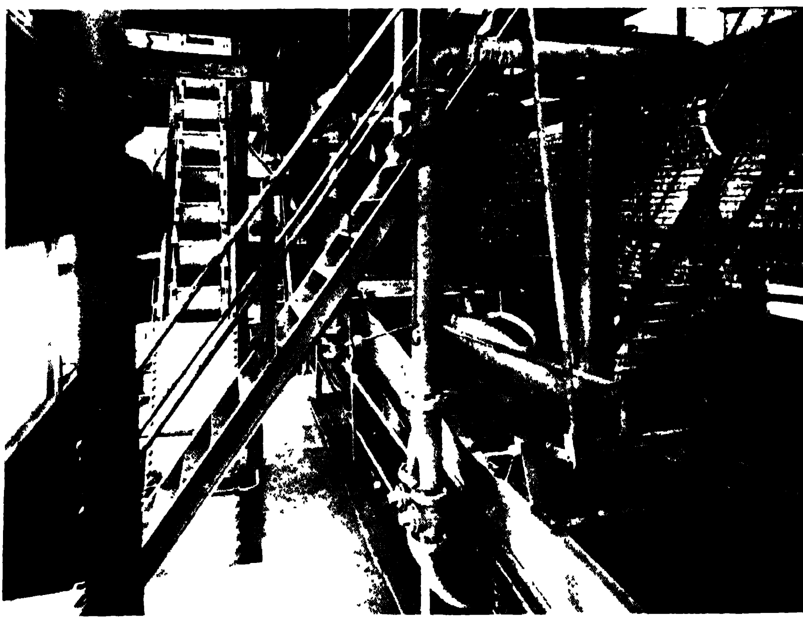


FIG. 110.—View of Main Washing Trough and Rewash Elevator, Rheolaveur Nut-Coal Washery.

[To face page 273.

narrower than the boxes, K and L, in the main trough, and the main trough discharge boxes require the admission of water at two points to ensure the proper and even spread of water across the width.

The total power provision is 135 h.p. made up as follows :

	h.p.
Raw coal conveyor	10
Washed coal mixing conveyor	10
Main water circulating pump	40
Auxiliary pump for sprays	5
Main driving motor	50
Small coal installation motor	20
Total	135

The pump motors and raw coal and washed coal conveyor motors are direct-coupled.

The main driving motor supplies power for the rotating screen, G (Fig. 109), the jigging screen, Q, the rewash elevator, R, the shale, and middlings elevator, O, the shale scraper conveyor, P, the crusher, S, and the flap valves eccentric. The small coal installation motor drives the worm conveyor, I, and the elevator, W.

For a washer with a capacity of 75 tons per hour, the power provision of 135 h.p. is not high, although the washery contains several items that would not normally be necessary in a nut-coal washery, viz., the crusher (20 h.p.), the small coal jigging screen (6 h.p.), and the mixing conveyor (10 h.p.). The height to which the shale and middlings must be elevated for rewashing is excessive, and was necessitated only by the height above the ground of the existing refuse bunker (*vide* sectional elevation, Fig. 109). Without these extra items the horse-power would be reduced by 40 h.p.

The second floor of the washery is illustrated in Fig. 110, which shows the main washing trough and the rewash elevator, elevating the discharge from the second rheo-box in the main trough back to the head of the trough. Attached to the trough, in the right foreground of the photograph, may be seen the mechanism operating the opening and closing of the flap-valves in the rheo-boxes and the double water supply to the underside of the boxes in the main trough.

The results of washing are given in Table 83, these results referring to samples which we collected over four shifts.

These results give rise to satisfaction on two accounts. Firstly, in every case, the ash content of the refuse is high, indicating a high yield of recoverable coal. Secondly, the washed coal fractions and the refuse are of reasonably uniform ash content from shift to shift suggesting that the washer can be relied upon to give regular washing.

TABLE 83.—RESULTS OF WASHING

	Shift A. Ash p.c.	Shift B. Ash p.c.	Shift C. Ash p.c.	Shift D. Ash p.c.
Raw coal, 3½ to 0 in.	—	—	16·8	19·1
Washed coal, 3½ to ⅜ in.	8·4	8·0	7·8	8·8
,, ,, ⅜ ,, 0 ,,	10·3	11·1	12·6	11·9
Refuse, 3½ to 0 in.	72·9	69·4	76·3	72·4

The irregularity of the feed during one shift, both as regards ash content and size distribution, is shown in Table 84, which gives particulars of hourly samples of the raw coal collected during shift C, each sample consisting of about 1 cwt.

TABLE 84.—ANALYSIS OF HOURLY SAMPLES OF RAW COAL

Time	6.30	7.20	8.10	9.15	10.15	11.15	12.15	1.15	—
In.	Percentages.								Mean.
3½ to 1¾	29·2	36·7	37·5	47·5	42·1	31·7	37·7	46·7	38·6
1¾ ,, 1	36·6	35·4	36·1	29·7	27·0	33·9	35·0	26·1	32·5
1 ,, ¾	20·7	12·6	17·4	15·1	18·5	16·9	14·1	16·7	16·7
¾ ,, 0	13·5	15·3	9·0	7·7	12·4	17·5	13·2	9·3	12·2
Ash content of bulk sample	17·5	9·4	20·3	15·8	18·6	12·6	18·8	21·4	16·8

The float and sink analysis of the raw coal, is given in Table 85.

TABLE 85.—FLOAT AND SINK ANALYSIS. Raw Coal

S.G.	3½–1¾ in.	1¾–1 in.	1–¾ in.	¾–½ in.	½–¼ in.	¼–0 in.	Total.
< 1·3	18·4	16·4	6·2	5·0	1·6	0·6	48·2
1· –1·4	6·7	6·5	3·8	1·6	0·3	0·2	19·1
1·4–1·5	4·8	2·1	2·7	0·5	0·2	0·1	10·4
1·5–1·6	1·8	1·2	0·6	0·2	0·1	0·1	4·0
1·6–1·8	5·2	2·4	0·3	0·4	0·1	0·1	8·5
> 1·8	2·0	3·8	2·9	0·9	0·1	0·1	9·8
Total	38·9	32·4	16·5	8·6	2·4	1·2	100·0

From Table 85 it is apparent that 22·9 per cent. of the raw coal had a density between 1·4 and 1·8. Much of this material was col-

lected at the first rheo-box, separated from true shale in the shale rewashing trough and crushed prior to rewashing. The washed coal was found to contain about 5 per cent. more "coal" particles than was present in the whole of the raw coal, representing 15 to 20 tons per shift of saleable coal recovered by crushing, which would otherwise have been lost.

Comprehensive float and sink tests were performed on the samples from shifts A and C only. The results for the refuse are given in Tables 86 and 87. The weights of refuse submitted to the float and sink tests were: Shift A, 48 lb.; shift C, 124 lb.

TABLE 86.—FLOAT AND SINK TEST. Refuse per cent. Shift A

S.G.	$3\frac{1}{2}$–$1\frac{3}{4}$	$1\frac{3}{4}$–1	1–$\frac{3}{4}$	$\frac{3}{4}$–$\frac{1}{2}$	$\frac{1}{2}$–$\frac{1}{4}$	$\frac{1}{4}$–$\frac{1}{16}$	$\frac{1}{16}$–0	Total.
< 1·5	Nil	Nil	Nil	Nil	0·02	0·01	0·01	0·04
1·5 1·6	Nil	Nil	Nil	0·04	0·07	0·02	0·01	0·14
> 1·6	4·30	15·60	13·40	17·86	43·81	3·87	0·08	99·82
Total	4·30	15·60	13·40	17·90	43·90	3·90	1·00	100·00

TABLE 87.—FLOAT AND SINK TEST—Refuse per cent. Shift C

S.G.	$3\frac{1}{2}$–$1\frac{3}{4}$	$1\frac{3}{4}$–1	1–$\frac{3}{4}$	$\frac{3}{4}$–$\frac{1}{2}$	$\frac{1}{2}$–$\frac{1}{4}$	$\frac{1}{4}$–$\frac{1}{16}$	$\frac{1}{16}$–0	Total.
< 1·5	Nil	Nil	Nil	Nil	Nil	Nil	Nil	Nil
1·5–1·6	Nil	Nil	Nil	Nil	0·9	0·2	0·1	1·2
> 1·6	13·0	22·0	20·8	12·0	26·0	4·0	1·0	98·8
Total	13·0	22·0	20·8	12·0	26·9	4·2	1·1	100·0

These figures are noteworthy, and demonstrate in a surprising manner the highly efficient removal of coal from the shale in the rewashing trough. They are all the more satisfactory when it is remembered that, allowing for the crushing of middlings, the coal in the washer contains mixed sizes from $3\frac{1}{2}$ in. to 0, and that the washing is accomplished in two troughs only.

THE RHEOLAVEUR SLURRY WASHER

The Rheolaveur slurry washer is, in effect, a simplified fines washer. A view of the slurry washer erected for Messrs. The South Yorkshire Coking and Chemical Co., Ltd., is shown in Fig. 111. A slurry washer may consist of either three or four washing troughs about 80 ft. long but only 12 to 15 in. wide. The arrangement of the troughs is the same as in the normal fines washer, except that in each

of the two upper troughs the first washing section is stepped down to a second section of gentler inclination. The essential difference

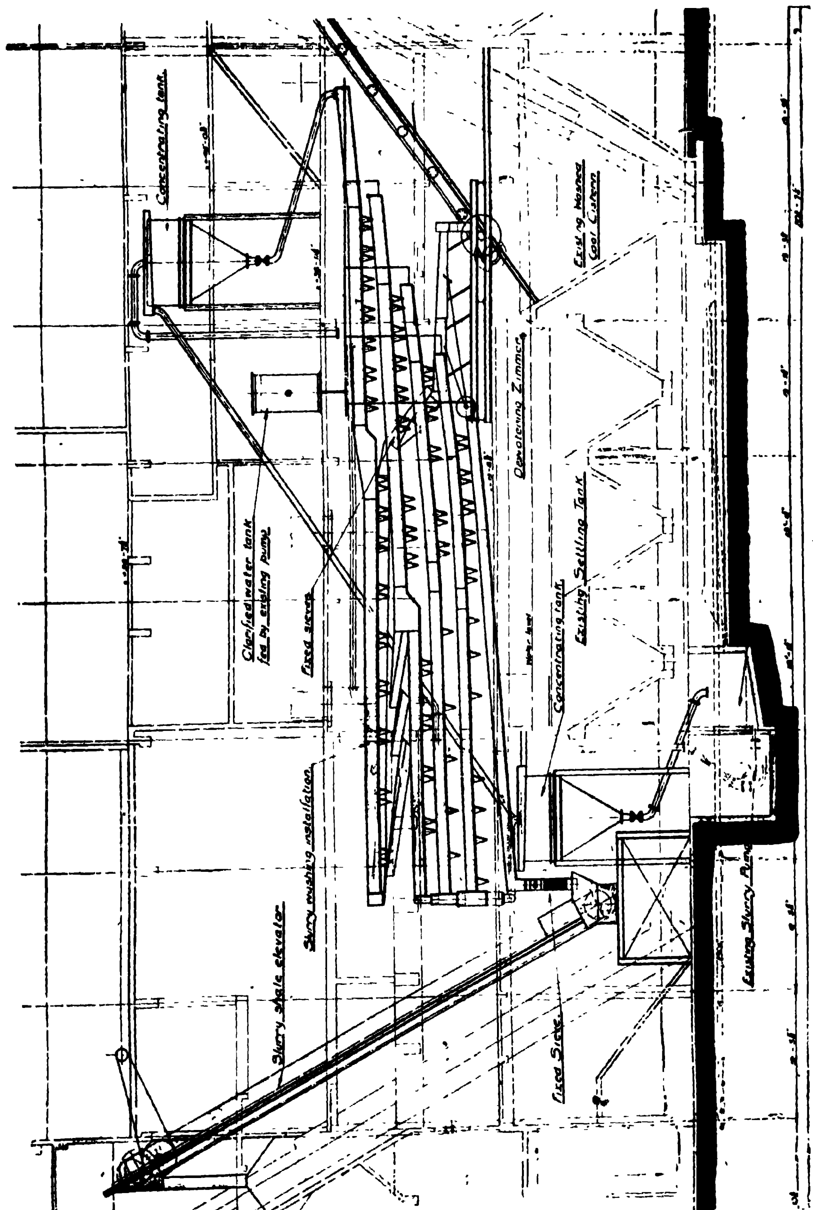


FIG. 111.—Elevation of Rheolaveur Slurry Washer.

between the fines washer and the slurry washer is that, in the latter, no upward water currents are supplied to the rheo-boxes, since only slow currents are used, and the discharge from each rheo-box is not

sufficient to cause vortex currents. The washed product is only taken from the first trough (instead of from the first two as in the fines washers) and so a greater percentage of the material is rewashed.

The actual rheo-boxes used in a slurry washer are therefore of much simpler construction, and have no central partition or water cocks. Below the discharge orifice a circular orifice plate is fitted with four holes for the final discharge of material from the rheo-box. The circular plate is also fitted with three loops into which a small bar may be inserted to rotate the orifice plate if one of its holes has become blocked.

The slurry supply may be taken from the bottom of a settling sump, or spitzkasten, and elevated to the head of the washer, where it is thickened in a concentrating cone and then passed to the washing troughs. As usual, a certain amount of material is returned to the head of the first trough for rewashing.

The washed slurry is passed on to a "zimmer," or shaking screen, with copper gauze covering, and dewatered. A spray of clean water is used at the shaking screen to remove any fine clay material deposited on the washed slurry from the washing water. This enables dewatering to be more efficient, and a product containing about 30 per cent. of water is obtained. The rejected dirt is added to the dirt from the fines washer.

Andry (*Proc.* S. Wales Inst. Eng. 1926, 41, 567) records the results given in Table 88 for the working of a slurry washer in the Sarre coalfield during a period of eight days.

TABLE 88.—RESULTS OF WASHING SLURRY.

Size (in.).	Raw Slurry.		Washed Slurry.		Refuse.	
	Weight per cent.	Ash per cent.	Weight per cent.	Ash per cent.	Weight per cent.	Ash per cent.
$> \frac{1}{6}$	0·26	13·3	0·06	21·6	—	52·1
$\frac{1}{6}-\frac{1}{10}$	2·94	13·3	0·34	4·5	0·29	47·2
$\frac{1}{10}-\frac{1}{25}$	9·52	12·8	7·69	3·7	8·64	58·0
$\frac{1}{25}-\frac{1}{30}$	11·02	23·8	27·94	3·9	12·42	59·5
$\frac{1}{30}-\frac{1}{40}$	5·18	25·1	15·64	3·7	12·11	60·0
$\frac{1}{40}-\frac{1}{60}$	21·56	40·8	34·52	7·9	46·76	63·6
$\frac{1}{60}-\frac{1}{250}$	43·03	64·1	13·09	21·4	18·57	68·1
$\frac{1}{250}-\frac{1}{400}$	1·44	62·7	0·17	32·6	0·36	67·6
$< \frac{1}{400}$	5·05	58·9	0·55	36·5	0·83	66·1
Total .	100·0	45·8	100·0	8·1	100·0	63·0

Of the raw slurry, 87·3 per cent. was less than $\frac{1}{25}$ in. in size, and for a raw material of this nature the results are satisfactory,

It will be observed that the washed slurry greater than $\frac{1}{40}$ in. contains less than 4 per cent. of ash.

The slurry washer at Glasshoughton, Yorks, gave the following average results in October, 1927 :—

	Per cent. Ash.
Raw slurry	24·0
Washed slurry	8·1
Refuse	64·1

On one day, the ash content of the washed material was 5·7 per cent. and of the refuse 69·8 per cent. The possibility of washing slurry to this low ash content is of great importance to the coking industry, for clean slurry can be drained far more easily than dirty slurry, and, in the ovens, clean slurry has little harmful effect on the quality of the coke produced if well admixed.

General

Experience with Rheolaveur washers has shown that they are capable of many variations in design to suit particular types of coal. This elasticity of design has proved to be a very valuable feature, although, at first sight, it might give an impression of complication. From the general experience gained with the use of Rheolaveur washers of varied design, and dealing with many different types of coal, some general modifications of design have become established practice. These general modifications, which have proved to be real improvements, have not yet been incorporated in many English plants on account of the difficulties which have affected the coal trade in the last few years. On the Continent, however, many plants have been erected with modified and improved design, such, for example, as the addition of the fourth trough to the fines washer and the use of a rewashing trough in the nut-coal washer.

The general control of a Rheolaveur washer is effected by adjusting the quantity of material rejected from the various rheo-boxes until a washed coal of the desired ash content is obtained. The washery is usually constructed so that it is possible to produce coal with the desired ash content when working with some of the rheo-boxes closed. If, subsequently, a cleaner product is required, the closed boxes can be put into operation. The rewashing to which the rejected material from the rheo-boxes in the first trough is subjected is sufficient to ensure that the final refuse contains a high percentage of ash. The control of the composition of the washery products of a Rheolaveur washer is, therefore, an easy matter, and the products can readily be altered to suit market requirements. For example, a yield of 75 per cent. of coal containing 5·0 per cent. of ash could be raised to a yield of, say, 80 per cent. of material containing 7·0 per cent. of ash if this were a readily saleable product, the adjustment of the washer being altered to make a separation

at a higher specific gravity. The Rheolaveur washer recently installed at the Loomis Breaker of the Glen Alden Coal Co. (U.S.A.) for anthracite washing makes a separation at S.G. 1·95, the clean anthracite from $3\frac{7}{16}$ to $\frac{3}{64}$ in. containing only 1·2 per cent. of sinks at S.G. 1·95 and the refuse 1·7 per cent. of floats at S.G. 1.95. (The ash contents are respectively 7·7 per cent. and 80·2 per cent.) Thus the Rheolaveur washer is particularly suitable for a firm washing coal for sale.

Proof of its adaptability is provided by the fact that the Rheolaveur Washery Company give a guarantee that the ash content of the clean coal shall not vary by more than 1 per cent. on either side of a stated value, which value can be altered subsequent to erection. At the same time, the guarantee stipulates that the ash content of the refuse will be not more than 2 per cent. less than the theoretical maximum for any given clean-coal ash-content.

The water supply necessary with a Rheolaveur washer is lower than in simple trough washers, and may only amount to 650 gallons per ton of coal of 0 to $\frac{3}{8}$ in. size, for capacities from 60 to 150 tons per hour. For nut-coal washers, more water is circulated per ton of coal. and may amount to about 1,200 gallons per ton of coal for a capacity of 50 tons per hour. The average over-all water requirements of five typical Rheolaveur washers, dealing with coal of 0 to 2 in. or more, are about 1,260 gallons of water per ton of coal for capacities of 60 to 80 tons per hour, and about 920 gallons per ton of coal for capacities of 125 to 150 tons of coal per hour.

The power necessary is mainly for water circulation. There is a saving compared with jig washers of the power necessary to give pulsations in the wash boxes, but a small amount of power is necessary for the rewash elevators. For eight typical Rheolaveur washeries with capacities varying from 50 to 150 tons of coal per hour, the average over-all power required was 0·95 h.p. per ton of coal. The power requirements are therefore low.

Considering the Rheolaveur washer in general, it has an advantage compared with the plunger jig type of washer in having no mechanically moving parts in the fines washer, and only the clack valves of the rheo-boxes in the nut-coal washer. In this respect it resembles the Baum type of jig, although, unlike all jig washers, it has no fixed screen plates which are liable to rupture. The liability for repairs is therefore reduced.

It has been stated in the past that the Rheolaveur washer, especially the fine-coal washer, is susceptible to irregular washing if the raw coal delivered to the washer varied in the proportions of coal and dirt. This was true with the earliest washers, in which inadequate provision was made for rewashing. In modern washers, the continuous circulation of a considerable proportion of particles of density intermediate between that of coal and shale, enables irregularities to be taken up without interfering with the efficiency of operation. If the proportion of dirt increases, more dirt is

returned to the trough for rewashing; if a greater proportion of coal is supplied, more coal is evacuated by the rheo-boxes.

In this respect the Rheolaveur has a distinct advantage over many other washers, for the irregularity is taken up automatically, and the washery man is not compelled to make adjustments to overcome the difficulty.

In the event of the temporary stoppage of the raw coal supply, the rheo-boxes discharging final dirt are closed and all the material in the washer is re-circulated until normal conditions are restored. No other present washer possesses this highly desirable quality; in all others the washer must be shut off and normal working conditions are not obtained until the expiration of five to fifteen minutes. In the fines washer, where there may be a number of rheo-boxes to close when the feed fails, it may take longer to throw the washer out of action than to shut off the air pressure and close the dirt slides in a Baum type of jig, but this is not a serious disadvantage, because it is not so necessary to shut off the washer immediately the failure of the feed is detected.

The Rheolaveur washer—in common with other trough washers—deals with thin pieces of flat shale more easily than do some washers using an upward water current or alternate upward and downward water currents. The gradual classification from trough to trough, and the large number of discharge orifices enable several products to be obtained if desired from a Rheolaveur washer, and for this reason it is especially suited to the treatment of coals of poor quality containing a large quantity of middlings. As previously pointed out, it can deal efficiently with all sizes of coal, and it appears to be able to wash slurry almost as easily as it washes fines. It is possible to obtain a cleaner over-all product more easily than when using any other type of washer as the sole means of washing, because no other washer capable of dealing with nuts up to 4 in. in size is also capable of slurry washing. It is well known that the smallest size of coal (say, 0 to $\frac{1}{8}$ in. size) may contribute as much to the total ash content of washed fines as the coal of, say, $\frac{1}{8}$ to $\frac{5}{8}$ in. size, which usually constitutes the greater part of the fines. It is by dealing efficiently with coal of 0 to $\frac{1}{8}$ in. size (as well as with the larger sizes) that the Rheolaveur washer as a whole can produce a cleaner product than can most other types of washer.

A further important feature of the Rheolaveur washer is that it occupies less ground space than a jig type of washer. Ample length is necessary, but the width required is only small. This has been proved to be such an advantage in practice that on the Continent, where many old types of jig washer have been in use long past their useful age, it has been possible to erect the various troughs necessary for a Rheolaveur washery whilst the jig washer was working. In such cases it is common practice to use the elevators previously used for the raw coal and washed coal, as well as the spitzkasten, water supply pipes and motors. Where, on a restricted site, an old

washing plant proves to be incapable of dealing with the capacity which may be required in modern practice, a Rheolaveur washer to deal with the necessary output could be erected more readily than any other type of washer without involving any loss of output through closing down the washing plant.

CHAPTER XV

CONCENTRATING TABLES: DEVELOPMENT

In addition to trough washers, which have been described in the two previous chapters, there are a number of other appliances for the separation of particles of different specific gravity which depend for their action upon a current of water flowing down an inclined plane. With the exception of trough washers, appliances dependent upon the phenomenon of alluviation have not been extensively applied to coal cleaning, though they have found a wide field of employment in the kindred art of ore-dressing. Strakes, frames, tyes and buddles were primitive appliances of this class, and the modern concentrating table was evolved from them by stages. The first stage was the invention of the endless-belt washer, introduced in England nearly a century ago. From the endless belt the vanner was developed, and the concentration table was the outcome of the vanner. Concentrating tables differ somewhat in principle, however, from vanners and appliances dependent solely upon alluviation, because of the riffling of the surface upon which separation is effected. This difference will be described later.

In endeavouring to trace the development of the modern concentrating table, it is necessary to deal with appliances which were mainly used for ore-dressing, but some examples of each class of appliance have been used for coal cleaning.

ENDLESS BELTS

Brunton's Belt.—Brunton's belt, invented in England in 1844 was the first endless-belt washer (*Mining Journal*, 1847, March 20th), and the evolution of the concentrating table may, with some justification, be considered to have begun with it.

Brunton's belt consisted of an endless strip of canvas, covered with paint, and driven over two rollers mounted on a wooden frame, one of which was placed higher than the other, so that the belt was inclined longitudinally. The appliance is shown in Fig. 112.

The upper roller was driven by gearing to make the belt travel upwards at a speed of about 15 ft. per minute. Mineral pulp was fed on to the belt at a point about one-third of its length from the upper roller, and water was supplied in a stream at a point nearer to the roller. The light mineral matter was carried down the belt by the current of water, but the heavier material remained stationary relative to the belt and passed upwards with it. Near the top,

adherent impurity, such, for example, as light sandy particles, was removed by the stream of fresh water, and the cleaned ore product was discharged into a trough of water through which the belt passed. The use of a transverse inclination and an additional water current perpendicular to the longitudinal path of the feed was suggested by the elder Brunton in 1841, but does not seem to have been used in his son's invention.

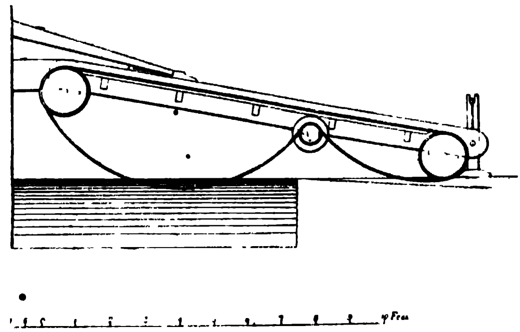


FIG. 112.—Brunton's Belt.

Although many appliances similar to Brunton's belt were introduced for various purposes, for example, the Hofmann, the Palmer, the Jones and the Harz belts for purifying metalliferous ores, and the Solway and the Beaussart belts, used in Belgium for calcium phosphate concentration (Schmidt, *Bull.* Soc. Ind. Min., 1894, 3, 8, 641), it was not until about 1880 that a similar appliance was used for coal cleaning, when the Rhum washer was introduced in Bohemia (*Oest. Zeit. für Berg. u. Huttenwesen*, 1884, 32, 532).

The Rhum Belt Washer.—The Rhum washer, shown diagrammatically in Fig. 113, was essentially an endless-belt appliance, similar to Brunton's, but the belt was entirely immersed in water. As originally constructed, it had a surface length of 1·35 metres ($4\frac{1}{4}$ ft.) and a width of 0·44 metres ($1\frac{1}{2}$ ft.), and moved horizontally in a tank of water.

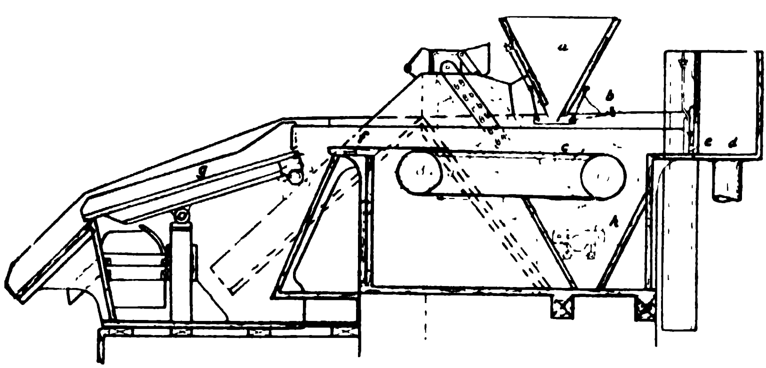


FIG. 113.—The Rhum Belt Washer.

The coal was distributed uniformly across the width of the belt, *c*, from a hopper, *a*, by a shaking feeder, *b*. A water current, admitted

from the box, *d*, through the gate, *e*, travelled along the trough in which the belt moved and carried the coal with it to the platform, *f*. The water current and the belt moved in opposite directions.

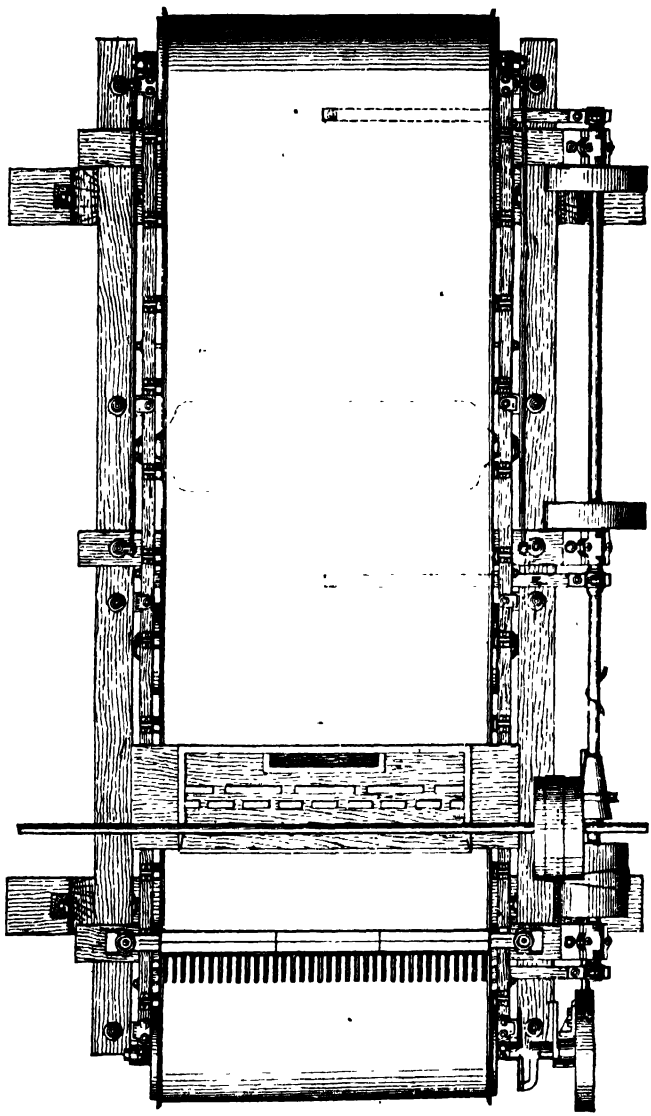


FIG. 114.—The Frue Vanner: Plan.

The heavier shale particles tended to adhere to the surface of the belt and were carried with it against the water current. The coal was discharged from the platform, *f*, on to a dewatering screen, *g*, and the refuse was discharged into the tank, *h*, at the other end of the belt, whence it was removed by a perforated bucket elevator.

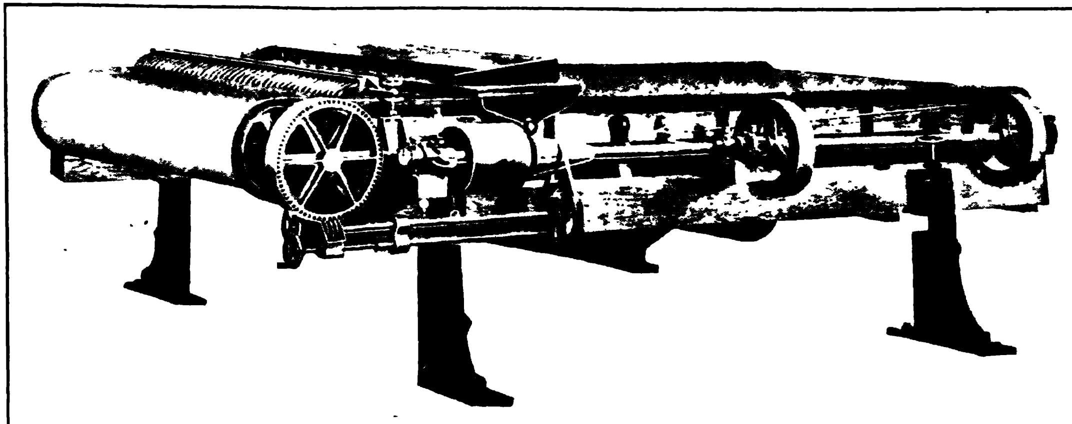


FIG. 115 View of Frue Vanner

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The Rhum belt had a relatively high capacity. According to Schmidt (*loc. cit.*) it could wash about 6 tons per hour of $1\frac{5}{8}$ to 1 in. coal, and about $3\frac{1}{2}$ tons per hour of $\frac{5}{8}$ to $\frac{5}{16}$ in. coal. A disadvantage, however, was the irregularity of its performance and the amount of attention required in operation. Slight differences in the speed of current or of movement of the belt, and irregularities in the quality or the rate of feed, interfered seriously with efficient operation, and neither as an appliance for coal-cleaning, nor for ore-dressing did it achieve any popularity.

The first vanner, or travelling belt with vibrations, was designed about 1860 by Hartwig (Henry, *Ann.* des Mines, 1871, 6, 19, 294). One disadvantage of all endless belt appliances (as well as of simple trough washers) was the liability for light particles to be mechanically entangled with the heavier material. This disadvantage was overcome to a certain extent in trough washers by the use of mechanical stirrers, and a similar object was achieved by causing an endless belt to vibrate. The shaking of the material on the surface of the belt loosened the bed and enabled a better separation to be effected.

On the Continent, the Hartwig vanner met with considerable success for ore-dressing, and, in 1867, a Hartwig machine was installed at Rossitz for the washing of coal. Its efficiency, capacity and precise operating details are not well known, and, in any event, it has been displaced by other similar appliances, the chief of which is the Frue vanner, invented and first used in the Lake Superior district in America, in 1874, and still used at the present time for the concentration of slimes.

The Frue Vanner.—The Frue vanner is shown in plan in Fig. 114 and a view is shown in Fig. 115. It consists of an endless rubber belt 4 ft wide, mounted on a frame, the rollers of which are about 12 ft. apart. Along its length the belt is inclined slightly so that the lower end is 3 to 6 in. below the upper end. The pulp is fed on to the upper end of the belt from a headboard which distributes it evenly across the width. The heavier ore tends to settle into a layer in contact with the belt, whereas the lighter material tends to occupy an upper layer resting on the heavier material. By rotation of the upper roller the belt is made to travel upwards at a rate of about 6 ft. per minute.

The vibratory motion is caused by the action of two eccentric pulleys on the frame supporting the endless belt. The eccentric pulleys are mounted on a shaft, which is belt-driven from a worm, the worm being actuated by gearing on the upper roller (Fig. 115). The average length of the stroke imparted by the eccentrics is about 1 in., and about 180 to 200 oscillations are made per minute.

The precise operating details, such as the slope, quality and rate of feed, depth of bed, rate of oscillation and rotation of the rollers depend upon the nature and size of the material to be treated. For successful operation, a uniform depth of bed, usually about ten

particles thick, must be maintained. If the bed is too thin, the capacity is reduced, if too thick, the separation is inefficient. The Frue vanner treats about $\frac{1}{5}$ to $\frac{2}{5}$ cubic foot of pulp per minute (about $\frac{1}{4}$ ton of ore per hour) and requires about $\frac{1}{2}$ h.p. for the drive.

Miscellaneous Vanners.—The oscillations imparted to the frame of a Frue vanner are sideways (transverse to the direction of the water current and the direction of movement of the belt). Vanners with an " end-shake " (an oscillation parallel to the water current and the movement of the belt), as distinct from " side-shake " of the Frue, have also been used, chiefly in America. Such are the Triumph, the Woodbury and the Embrey. End-shake vanners have been stated to be better for fine material and side-shake vanners for coarse material. With all vanners, the object of the oscillation is to agitate the particles, since a simple shaking action materially assists the formation of two layers, the one of heavier material on the bottom and the other of lighter material above it. The actual separation of the two layers so produced is essentially similar to that effected on Brunton's belt.

Another form of vanner has been used (*e.g.*, the Ellis and the Snyder concentrators), in which the belt surface has a gyratory motion imparted to it by an unbalanced, high-speed flywheel, but in this form, vanners do not appear to have been extensively used.

A number of other appliances incorporating an endless belt as the washing surface have been employed, in which the water current was transverse to the direction of motion of the cloth, and these appliances are the real precursors of the modern shaking table.

Vanners have had an extensive use for the concentration of fine sands and slimes, but since the introduction of the modern concentrating table, and the still more extensive application of froth flotation to the concentration of fine particles, their extended use has practically ceased. They have been described in view of their historic interest.

CONCENTRATING TABLES

On a vanner, the oscillation is a regular to-and-fro motion with the same speed in each direction, and its purpose is to keep the bed loose and enable mechanically-entangled light particles to be freed from the lower portion of the bed. At the same time it assists the formation of layers of material of different specific gravities.

On a modern shaking table, however, the reciprocating movement of the deck is not a regular to-and-fro movement with the same speed in each direction. The forward movement of the surface is terminated suddenly and the direction of motion is rapidly reversed. It is therefore more of a jerking motion, and it not only enables stratification to be more complete, but it has the further effect of assisting the motion of the particles in their passage to their points

of discharge. On modern tables the motion is transverse to the direction of flow of the water, and when the raw material is fed in bulk it spreads out, as it were, in fan shape, its path being the resultant of two forces acting at right angles to each other, one the water current, and the other the impetus given by the oscillation of the table. Such tables, with the water current and the jerking motion perpendicular to each other, may be called "side-jerk" tables.

Some of the older tables were not of the side-jerk, but of the "end-jerk" type, the direction of the jerking motion being parallel to that of the water current. Under these conditions, the main benefit of the vibration of the table was in assisting stratification of the material, but, because the oscillation was of a jerking nature, it was effective in transporting the heavier material, which clung to the surface, towards its discharge point.

End-jerk tables have the disadvantage that they separate the pulp into only two products ; the heavy concentrates collect at the upper end of the table, and the lighter material is discharged in a stream of water at the lower end. Side-jerk tables spread out the material, and any number of products may be collected according to the number of the discharge collectors. Some of the older end-jerk tables had the further disadvantage that they were not continuous in operation, and the accumulation of concentrates required to be removed periodically. For these reasons they have been almost entirely superseded by side-jerk tables. Nevertheless, they are of especial historical interest in coal-cleaning practice, because two such tables were employed in England and America for washing coal. These were the Campbell and Craig tables.

The Effect of a Jerking and Reciprocating Movement of the Surface.—Before describing the construction and operation of concentrating tables, it is essential that the effect of the jerking motion of the deck be appreciated. Its effect may be described in the following way :—

If a particle rests on a horizontal plane which is made to move slowly to the right and then to the left, the particle will remain in contact with the plane and move backwards and forwards with it, provided that the frictional forces between the particle and the plane exceed the momentum acquired by the particle. If, however, the plane, with the particle resting on it, moves to the right with a gradual acceleration, the particle gradually acquires a relatively high momentum. If the plane be suddenly arrested, the momentum of the particle will exceed the force of friction, and the particle will continue to move forward. Its speed will gradually decrease until it comes to rest at a point to the right of its original position on the plane. Whilst the particle is still in motion to the right, the plane may be made to travel back to its original position. If the motion of the plane in the reverse direction be rapid at first, so that the

change from its forward to its backward movement is accomplished by a sudden jerk, the plane is, as it were, withdrawn suddenly from underneath the particle, and the tendency for the momentum of the particle to exceed the frictional resistance is increased. By combining and repeating these two movements, the particle can be made to travel across the plane for any required distance.

Suppose that the plane be inclined at an angle to the horizontal: the force required to move the particle down the plane will be less than that required to move it up the plane. If, therefore, there are two particles on the plane and the reciprocating movement of the plane affects one more than the other, one particle can be made to move up the plane by its jerking movements, whilst the other remains relatively stationary or travels downwards. In practice, when a stream of particles is moved over a jerking plane, the bed of material is kept in a loose condition and the heavier particles tend to settle into a lower layer in contact with the plane. They are therefore affected to a greater extent by the motion of the plane than the lighter particles above them. It has been shown in Chapter XII that, if a stream of water be made to flow down a plane covered by a stratified mass of mineral, the current of water affects the lighter and upper layer to a greater extent than the heavier and lower layer, tending to carry the lighter particles downwards, leaving the heavier particles behind.

On a shaking table, the two effects are combined, the reciprocating movement of the surface tends to displace the heavier particles in one direction, the current of water tends to displace the lighter particles in another direction, and by this means a separation of the two types of material is readily effected. The chief difference between end-jerk and side-jerk tables is that, in the former case, the jerking motion of the surface is parallel to that of the water current, and moves the heavier particles in a direction opposite to the current, whereas on a side-jerk table the jerk is perpendicular to the water current and tends to move the heavy particles in a direction at right angles to it.

End-jerk tables were first introduced in Western Germany, Austria and Hungary for the dressing of mineral ores. One of the oldest is the Salzburg table, consisting of an inclined deck suspended at each end. By means of a cam, the table was made to swing slowly in the direction of slope of the table against the action of a spring. The spring then caused a rapid recoil, which was terminated sharply by a bumping block. As a result, the heavy concentrates and middlings travelled up the surface and collected at the upper end, whilst the tailings were discharged in a stream of water at the lower end. The disadvantage, that it was not continuous in operation, was removed in later appliances.

The Campbell Coal Washing Table.—The Campbell table (Phillips, Eng. and Min. *Journ.*, 1893, 55, 128, and Claghorn, *Trans.*

Inst. Min. Eng., 1902, 23, 435), was the first concentrating table to be used for coal-cleaning. It was first installed in America about 1890.

Campbell tables are still in use in America at Johnstown, Pa., and Wehrum, Pa. The Rosedale washery of the Cambria Steel Company, at Johnstown, Pa., contains seventy-two Campbell tables with a total capacity of 400 tons per hour.

The table is illustrated in Fig. 116. It consisted of a shallow rectangular box, A, about 10 ft. long and 2 ft. 6 in. wide, suspended by four vertical rods (not shown in the diagram) which permitted a swinging motion. The base of the table was made in two portions;

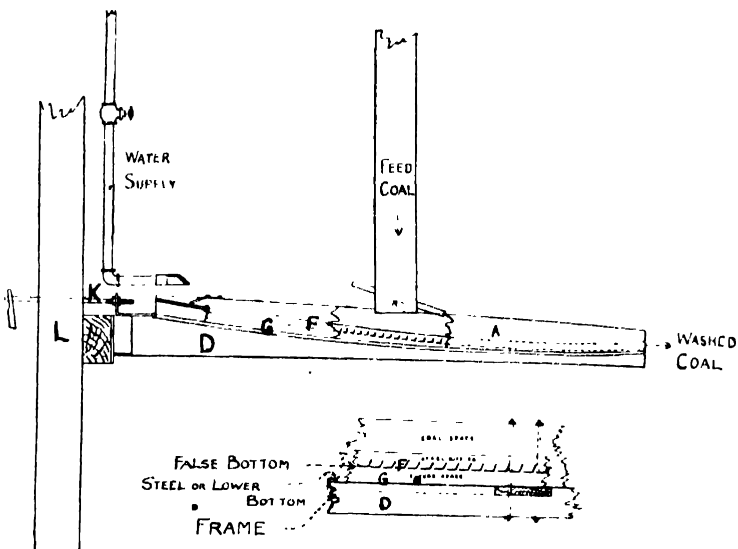

FIG. 116.—The Campbell Coal-Washing Table.

the lower or true bottom consisted of a steel plate, G, secured to the sideboards and the frame, D, and above it was a false bottom, F, resting on supporting strips. The two decks were about $1\frac{1}{2}$ in. apart, and each was curved (as is shown in the diagram) the amount of curvature adopted being found by trial. The false bottom, F, was a series of steel plates bent to form riffles and spaced $\frac{1}{32}$ to $\frac{1}{16}$ in. apart.

The vibratory movement of the table was accomplished by means of an eccentric and cam-lever working about a fixed fulcrum, the motion being communicated to the frame of the table by a series of connecting rods, K. The arrangement of the eccentric and the lever was so designed as to impart to the table a slow forward movement (in Fig. 116, towards the right), ending smoothly, and a more rapid backward swing (in Fig. 116, towards the left). The backward swing

was terminated abruptly by the bumping block and post, L. By these means the table was suddenly made to change its backward movement for a forward movement, and those particles which were in contact with the floors of the table tended to continue moving in the backward direction (left), whilst the table suddenly jerked away from them in a forward direction (right).

The coal was fed regularly from an overhead storage bin through a shoot on to the upper deck of the table at a point near the middle. Some of the washing water reached the table with the coal, the remainder being supplied from a distributor set transversely at the head of the table, but not attached to it.

In operation, a bed of refuse several inches thick was allowed to collect, so that it completely covered the riffles or false bottom. The raw coal tended to separate into layers, with the coal uppermost. The flow of water down the plane carried the coal towards the lower discharge end, whilst the jerking action of the table caused the movement of the refuse (in contact with or near the vibrating surface of the table) towards the upper discharge end. The false bottom of the table enabled small particles of shale and pyrites to work their way through the bed of refuse on the riffles, and to pass through the gaps between riffles to the true bottom, where, being affected by the vibration of the table, they gradually worked their way to the refuse-discharge point.

The Campbell table had a capacity of 5 to 7 tons of small coal per hour, and required about ½ h.p., the water consumption being about 250 gallons per ton of coal treated.

A series of tests, involving the cleaning of 5,000 tons of coal, was conducted in 1900 for the Lackawanna Iron and Steel Company. The raw coal was screened to pass through bars spaced ¾ in. apart, and the distribution of the coal was as follows :—

In.	Per cent.
¾ to ½	27
½ „ 5/16	42
5/16 „ 0	31

The results of washing are shown in Table 89 :—

TABLE 89.—WASHING RESULTS (CAMPBELL TABLE)

	Per cent. of Feed.	Ash per cent.	Sulphur per cent.	Floatings at S.G. 1·4.	Sinkings at S.G. 1·4.
Raw coal	100·0	5·8	1·91	93·39	6·61
Washed coal	93·9	4·8	0·86	99·04	0·96
Refuse	6·1	45·0	17·62	1·47	98·53

The coal used in these tests had a low initial ash content, and the main object was the removal of sulphur. Since the sulphur content was reduced by half by the removal of only 6 per cent. of refuse, the pyrites responsible for it must have been present mainly as free particles, and not distributed through the mass of the coal substance.

Richardson (*Coal Age*, 1923, 23, 285), gives the results of five tests on the tables at the Rosedale washery, where, as stated, seventy-two Campbell tables are installed. The average result of these tests are as follows :—

	Ash per cent.		Sulphur per cent.
Raw coal . .	11·3	..	2·6
Washed coal .	7·8	...	1·3
Refuse . .	61·8	...	21·0

The Craig Coal-Washing Table.—The Craig table, previously used for gold washing, was used in England for coal cleaning about 1900 (Scott, *Trans.* Inst. Min. Eng., 1902, 23, 179). The principles

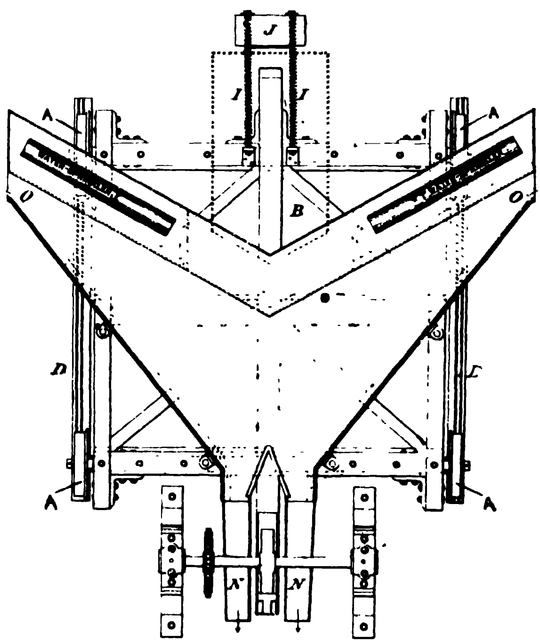

FIG. 117.—The Craig Coal-Washing Table.

of its action are similar to those of the Campbell table, but only one deck was used and the table was Y-shaped.

The table, illustrated in Fig. 117, was inclined downwards from the fork, B, towards the narrow end, with an inclination of about 1 in 6. The table was mounted on a framework supported by rollers, A, on an inclined track, D. The jerking motion was imparted

to the table by an eccentric, which pushed the table up the track. At the end of the slow forward stroke the table travelled backwards at an increasing speed, partly because of the tension of the springs, I, and partly because of the inclination of the track. The backward stroke was terminated suddenly by collision with the bumping block, J.

The raw coal was fed at B in a stream of water and a further supply of water was introduced by the sprinklers, P. It carried the cleaned coal down the table towards the narrow end, where it was discharged by the shoots, N. The jerking motion of the table resulted in the passage of the refuse up the table to the refuse-discharge shoots at O.

The Craig table treated coal from 1½ in. to 0, and with large coal had a capacity of 8 tons per hour. The length of the stroke was 5 in., and the table made sixty strokes per minute. Results of washing at Coanwood Colliery were given by Scott as follows:—

	Ash per cent.		Sulphur per cent.
Raw coal . .	11·50	...	1·96
Washed coal .	4·80	...	1·65

It is impossible to form any definite idea of the efficiency of the appliance from these results, for no mention is made of the yield or of the loss of coal in the refuse.

The Rittinger Table.—The Rittinger table was the first table using a side-jerk, that is to say, with the water current at right-angles to the direction of the jerking motion. It was 8 ft. long, 4 ft. wide, suspended by four rods, and actuated by a cam, spring and bumping post. The table was inclined along its length at an angle of 3° to 6°, the inclination being less, the coarser the pulp treated; a current of water flowed down the slope. The feed was supplied at one corner of the head of the table, being carried down it by the water current. Meanwhile the jerking action caused the heavier particles to travel across the table, so that separation into several products was possible at the discharge point.

The Rittinger table had a low capacity, required a certain amount of sizing of the feed, and was irregular in action.

The Wilfley Table.—Introduced in 1896 as an improvement on the Rittinger table, the Wilfley table is more extensively used to-day for ore-dressing than any other concentrating table; over 24,000 Wilfley tables having been installed in all parts of the world.

Its advantages over the Rittinger table were that: (1) it spread out the feed over a greater lateral range; (2) it had an improved mechanism for imparting the jerking motion to the table, which resulted in a more favourable separation of the feed and a reduction

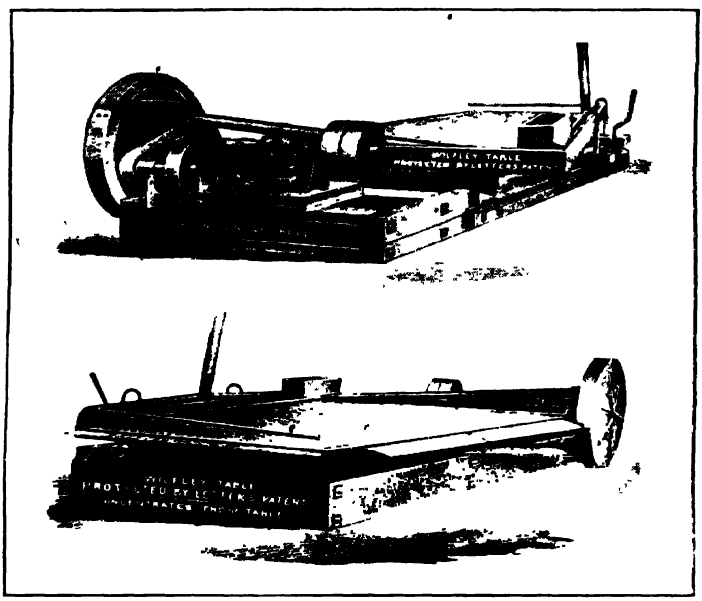

Fig. 118. The No. 3 Wilfley Table.

[*To face page* 293.

of the wear and tear of the machinery ; (3) it had a riffled surface, which considerably increased its capacity.

The spreading out of the feed enabled thinner beds of material to be used, thus increasing the probability of more accurate stratification of the feed according to density and size differences. There

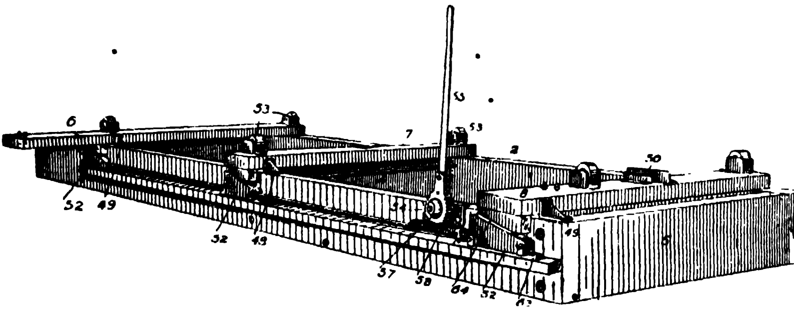

FIG. 119.—The No. 3 Wilfley Table : Framework.

was also the further benefit that, if desired, a larger number of products could be collected for subsequent treatment. The introduction of riffles on the surface not only increased the capacity of the table, but provided a number of opportunities for the rewashing of the heavy particles, and especially of middlings. Perhaps the

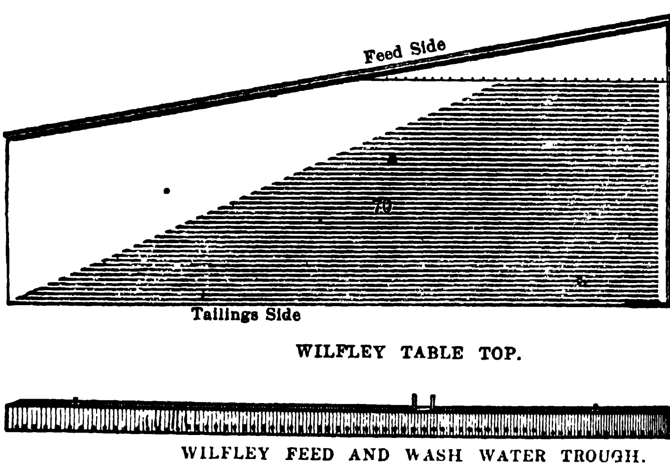

FIG. 120.—The No. 3 Wilfley Table : Deck.

greatest improvement, however, was the improved mechanism for creating the head motion. The old system of making the table hit a bumping post to destroy its momentum resulted in heavy wear of the machinery, and was especially destructive to the framework of the building, which took the whole shock of the bump. The improved mechanism obviated this damage to the building, and reduced the wear and tear of the framework of the table. The smoother

movement obtained was equally effective in carrying out the purpose of the bumping action, and, indeed, was advantageous in that it was more gradual and assisted stratification of the material without any tendency to disturb the stratified layers by creating a commotion on the deck.

The first Wilfley table was erected by Arthur R. Wilfley at his mill in Kokomo, Colorado, U.S.A. This, and the second model, were largely experimental, and the first really successful model was the

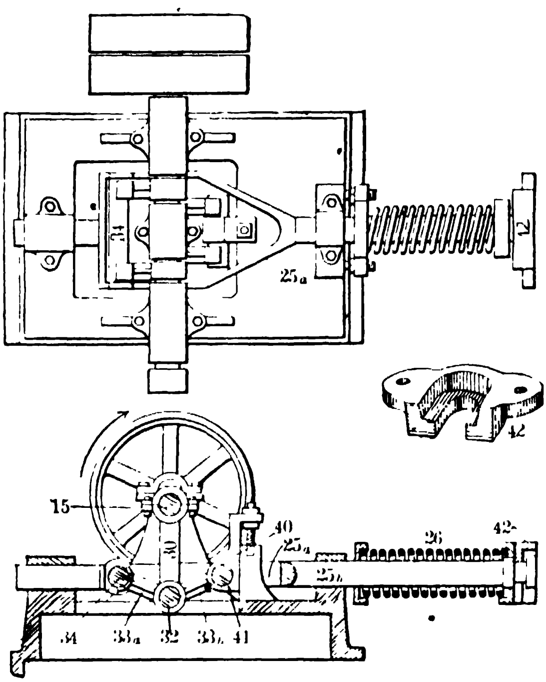

FIG. 121.—The No. 3 Wilfley Table: Driving Mechanism.

No. 3 Wilfley, introduced in 1898, and illustrated in Figs. 118, 119, which are reproduced from an 1898 catalogue.

The deck of the No. 3 Wilfley table was supported on three cross-timbers (6, 7 and 8, Fig. 119), borne on a framework consisting of two heavy longitudinal timbers strutted with four cross-pieces, and held together by tie bolts. The cross-timbers were screwed loosely to the frame on the tailings side of the table, and were supported at the feed side by three wedges, 52, by means of which they could be raised or lowered simultaneously, by the operation of a lever, 55, attached to a pinion, 54, working in a rack, 58. Six iron rollers on the cross-timbers, 53, supported the deck of the table. Side-play was prevented by a projection under the table working in a guide, and by connection with the actuating mechanism at the head of the table.

The surface, or deck, of the table (Fig. 120) was made of wood covered with linoleum and protected on the underside by steel plates. A series of riffles, each ½ in. wide and ⅞ in. apart, were fastened to the surface. At the mechanism end the riffles were ¼ in. high, and tapered down to nothing towards the concentrates end. Only the bottom riffle, which was nearly 16 ft. long, stretched right across the table; those higher up the table were shortened progressively in such a manner as to separate the table into two portions, one riffled and the other plain. The upper riffle was only 4 ft. long. The table was inclined slightly downwards from the mechanism to the concentrates end (½ in. in 16 ft.). An extra inclination of 1 in 4 was introduced at the mechanism end, the height rising to ¾ in. in the last 3 in. to prevent the formation of a bank of material.

The table was driven by the mechanism shown in Fig. 121. It consisted of a crank arm (or pitman), 30, working on an eccentric, 15. The crank arm was connected to a toggle, 33*a* and *b*, which communicated the movement of the crank arm to a yoke, 25*a*, attached by a connecting rod, 25*b*, to the table by means of a slotted keeper, 42. The motion was balanced by connecting a second toggle, 33*a*, to the movable rod, 34. As the crank arm rose the table moved to the left, and the spring, 26, was compressed; as the crank arm fell, the spring was released and the table moved smoothly back to its original position. This mechanism gave an accelerating forward stroke of the table, and a retarding backward stroke. The spring merely served to give a smooth movement.

For this reason it was screwed only tight enough to take up play in the bearings and prevent rattling. Further tightening only increased the bearing friction. As a result of the motion of the surface, the heavy ore particles were carried from the mechanism end towards the concentrates end of the table.

The length of stroke was adjustable by means of the block, 40; the best average length of stroke was given by the makers as ¾ in., with ⅝ in. and 1 in. as the minimum and maximum respectively for ordinary ore-dressing work. The speed recommended was 240 strokes per minute.

The feed pulp was supplied from the partitioned box (Fig 120), running along the length of the feed side of the table, the length over which it was distributed being controlled by the position of the partition A supply of wash water was distributed along the whole remaining length of the box along the feed side of the table.

The action of the table was identical with that of the modern Wilfley. The agitation of the pulp tends to cause the heavier particles to settle to the floor of the table. In so doing they become trapped in the riffles, and are prevented thereby from progressing further towards the tailings side of the table. The lighter particles, on the other hand, tend to accumulate in the upper layers, and, meeting the current of water, are gradually washed over the riffles to the tailings end. The lighter they are, the greater is their tendency

to pass over the riffles and the nearer to the head, or mechanism end, are they discharged. Whilst they are being carried along over the riffles, they are in a state of constant agitation, so that ample opportunity is provided for a heavy particle carried along with the lighter particles to sink between two riffles.

Particles of intermediate density pass over the first few riffles, but are trapped later as the feed spreads out and the bed becomes thinner. The particles trapped between two riffles sink to the floor of the table, and the jerking action of the table tends to carry them across the table between the riffles and towards the concentrates end. As they move in this direction the height of the riffles decreases and the large light particles, which may initially have sunk to the bottom of the bed, are affected by the water current washing across the riffles, and are thereby carried over the riffle. They may or may not be then trapped between two further riffles nearer the tailings side where the riffles are higher.

When the particles come to the end of a riffle on to the finishing surface, the water current washes the lighter material from the heavier material and into the next riffle. The particles of the heaviest material are but little affected by the current of water, and pass straight over the finishing surface to the concentrates collector. Particles of middlings, however, are washed gradually towards the tailings side, but, because they are trapped in the riffles from time to time, they are also propelled across the table in the direction of the concentrates end. If they are large they are more likely to be washed over a riffle than if they are small, but if so, they are trapped by a subsequent riffle and again moved across towards the concentrates end. Small middlings particles are carried to the diagonal line separating the roughing (or riffled) area from the finishing area, and they then make their way down the table along the ends of the riffles. When washing coal, the tailings side is the clean coal-discharge side and the concentrates end is the refuse-discharge end.

It may be seen that the action of the table includes provision for the *gradual* removal of the light from the heavy mineral. No attempt is made to effect any sudden and drastic separation, and the success, on the score of efficiency, with which the table met on its introduction may be attributed to this and to the provision afforded for the constant rewashing of middlings. The main separation of large light particles is effected easily and at the outset. The particles which are more difficult to separate pass forward from riffle to riffle until, finally, if separation is possible, it is reasonable to suppose that it has been accomplished.

Since 1898 a number of mechanical improvements have been effected in the details of the Wilfley table, but its general design is unaltered. In place of the rectangular deck, the head or mechanism end has been set inwards (towards the concentrates end) and the feeding and supporting arrangements have been improved. The modern No. 15 Wilfley table, illustrated in Fig. 122, is supported

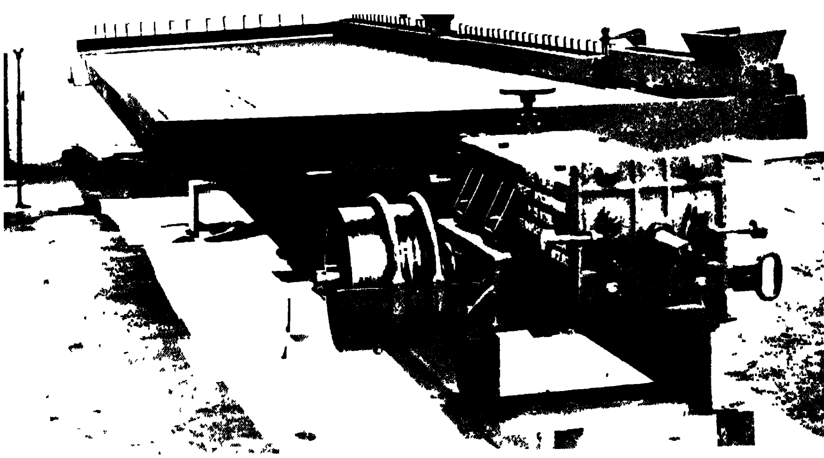

FIG. 122. The No. 13 Wilfley Table

on two longitudinal steel channels, on which two cross-channels are mounted. The inclination of the deck from the feed to the discharge sides is altered by tilting the cross channels.

The deck is supported by V-tread supports working in V-section tracks bolted to the cross channels. It consists of timbers laid diagonally and strengthened by wooden strips, the whole being kept rigid by side boards and iron stays. The upper surface of the deck is about 16 ft. by 6 ft., and is riffled in the same way as the earliest tables. The riffles consist of forty-six wooden strips, $\frac{1}{4}$ in. wide. The riffle nearest the tailings side is 13 ft. 6 in. long, and increases from a feather edge at the corner of the table to a height of $\frac{1}{2}$ in. (or more in special cases) at the head end. The other riffles increase from zero at the diagonal line along which they terminate to a height at the head *pro rata* to their respective lengths. The

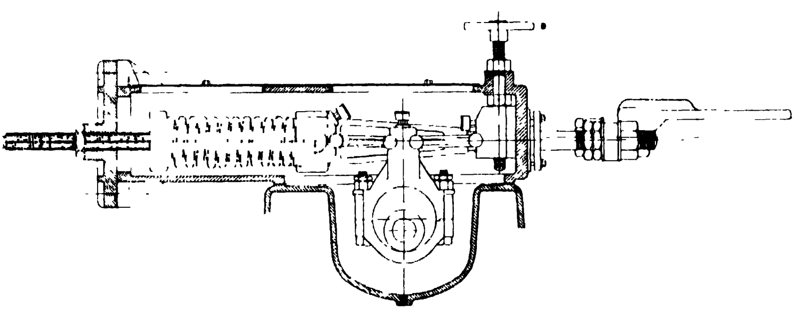

FIG. 123.—No. 15 Wilfley Table: Head Mechanism.

strips are usually placed $\frac{3}{8}$ in. apart. The upper riffle is about 4 ft. 9 in. long.

The pulp is fed from a box bolted to the side of the table, and the feed-box therefore vibrates when the table vibrates. Additional water is supplied along the whole length of the table by pipes attached to the ends of the cross channels.

The driving mechanism, shown diagrammatically in Fig. 123, consists of a toggle mechanism driven by a crank arm and an eccentric.

When the machine is in operation, the rear toggle is forced periodically against the rear toggle seat, drawing the thrust rod to the left (in Fig. 123). The thrust rod is supported in spherical bushes and connected to the deck of the table through the thrust rod head and a draw bolt on the under side of the deck. When the thrust rod is moved to the left, the coil spring is compressed, and when the crank arm has been further displaced and the thrust on the toggle is reversed, the spring ensures that the thrust rod moves smoothly towards the right. The right-hand toggle thrusts against a fixed pin in the adjusting block, and the precise length of the stroke imparted to the table depends upon the position of the block.

When the table is correctly set and adjusted, it runs without noise other than a gentle throb.

Only a relatively small adjustment of the stroke can be effected by raising or lowering the toggle adjusting block. A considerable change can, however, be accomplished by re-setting the eccentric. The length of stroke can be varied from ½ in. to 1½ in., and the speeds of oscillation recommended with different stroke lengths are as follows :—

Length of Stroke. In.	No. of Strokes per Min.
½	330
¾	300
1	270
1¼	240
1½	220

The adjustments, other than the side elevation, that can be effected for any particular purpose are the end elevation, the amount of water mixed with the raw material, and the length of stroke.

The Wilfley is a true pioneer in table concentration, and the broad principles of design of the Wilfley table are those accepted by all makers. It is interesting to see that, in the modern Wilfley table, any new features are only slight modifications of earlier design. The chief features, the termination of the riffles in a diagonal line across the deck, and the mechanical method of imparting impulses to the table, are unchanged after thirty years of experience.

CHAPTER XVI

CONCENTRATING TABLES—*continued*

THERE are certain features common to all coal-washing tables. All are wooden structures, approximately rectangular in shape, covered with wooden riffles, or, occasionally, brass, rubber or rubber-faced riffles, tacked on to a protective surface of linoleum or rubber. A driving mechanism is placed at one of the narrow sides (or ends) to impart to the table an accelerating forward motion, a sudden reversal, and a retarded backward motion. The refuse is discharged at the end remote from the head or mechanism end, and the cleaned coal is discharged along one of the longer sides. The middlings collect near the corner between the coal-discharge side and the refuse-discharge end.

The feed is supplied at the corner diagonally opposite to that at which the middlings collect, and washing water is distributed along the fourth or feed side.

The tables are all inclined transversely downwards to provide a water current from the feed side to the clean-coal discharge side, and usually there is a slight upward inclination from the head or mechanism end to the refuse-discharge end.

This upward inclination along the length of the table is to prevent water currents from moving towards the refuse-discharge end and carrying fine coal particles with them. The height of the refuse-discharge end is usually 1 to 2½ in. above that of the mechanism end, but 3 or 4 in. may often be required. This is not, however, always the case.

The Massco Coal-Washing Table.—The Wilfley table described in the previous chapter was designed primarily for the treatment of small particles of mineral ore. With small particles, the concentration was effected primarily on the roughing or riffled area, and, finally, on the finishing or unriffled area.

When it is desired to treat a feed which does not consist essentially of fine sands or slimes, this arrangement of riffling may not be the most satisfactory, and, in ore-dressing practice, the design of the deck of the Wilfley table is modified to suit special circumstances. Normally, the middlings product is collected at a corner of the table, the purified ore or concentrate being taken along one end. Such a division is suitable with a normal ore low in mineral content and easy to concentrate. When, however, separation is not so simple, because of the high grade of the ore, or because there is a considerable

proportion of material of intermediate density, it is necessary to alter the riffling system. For example, if there is a large middlings fraction, it is usual to arrange that three or four of the longest riffles, and not only the bottom one, end along the full length of the table.

In other circumstances, especially with a coarse feed, or an ore giving a high yield of concentrates, the riffles may be placed all over the deck. Under such conditions the valuable mineral matter is collected high up on the concentrates end of the table, and there is a wide section of the table containing middlings. The effect of such an arrangement is, normally, to increase the capacity of the table considerably, but, by eliminating the finishing area, to reduce

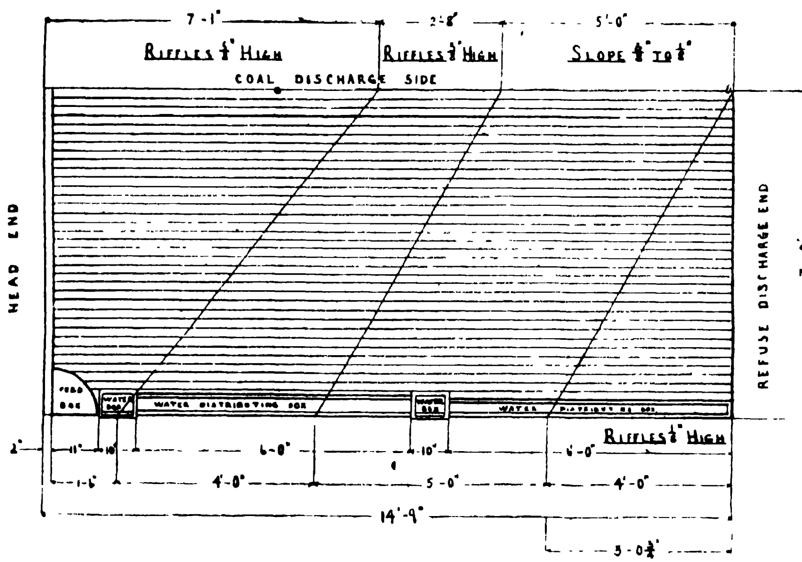

FIG. 124.—Deck of Massco Table.

somewhat its otherwise very high efficiency, especially with regard to the removal from the concentrates of small particles of impurity.

The very accurate separation required in, say, the dressing of auriferous sands, is unnecessary for coal-washing, for then the feed is of relatively low value, and the loss of a few particles of coal may be risked if, by so doing, the capacity of the appliance can be considerably increased. Consequently, for coal-washing, the deck of the standard Wilfley table is usually riffled all over. In these circumstances, the refuse is discharged from the ends of the uppermost riffles, and the whole of the remainder of the refuse-discharge end and the clean-coal discharge side are used to collect coal and middlings, the cleanest coal being found near to the mechanism end and the dirtiest coal near to the point where the refuse is discharged. Between these two positions there is a gradual increase in the ash

content of the coal, and by choosing a suitable point along the edge of the table at which to make the division between the clean coal and the middlings, a product can be obtained containing any desired amount of ash (greater than a certain minimum). This is illustrated by the results given later.

The first experiments in coal cleaning using a Wilfley table were made in 1906, the standard No. 5 table being used. These experiments led to the design of the Massco, which was the Wilfley coal-cleaning table. It was found, in practice, that, for coal washing, the dimensions of the standard Wilfley deck were inadequate, and the deck of the Massco table was made 14 ft. 9 in. long, but the width was increased to 7 ft. Moreover, the deck was made rectangular. In practice, the heights of the riffles, and the distances between them, is varied according to the size of the coal treated, but the standard Massco table has thirty-five riffles, all running right across the table, each $\frac{5}{16}$ in. wide and spaced $2\frac{1}{4}$ in. apart (between centres). The surface is divided into four areas, shown in Fig. 124, in which the heights of the riffles are different. Near to the head end, the riffles are $\frac{5}{8}$ in. high; in the second area they are $\frac{3}{8}$ in. high; towards the refuse-discharge end they slope regularly from a height of $\frac{3}{8}$ in. to $\frac{1}{8}$ in., at which minimum height they continue to the refuse end.

The feed is distributed from one corner of the table, as shown, and not over a considerable length along the feed side, as on the Wilfley table. The upper, or feed, side, is occupied by two water-distributing boxes, which supply the washing water. The coal itself is supplied into the feed-box with about twice its own weight of water, the actual proportions varying from 1·7 to 2·5 of water to 1 of coal according to the size of the particles.

Massco tables are used for the treatment of unclassified material with a maximum size of about $\frac{1}{4}$ in. Under these conditions they deal with between 4 and 8 tons of raw coal per hour, the capacity being dependent upon the relative proportions of coal, middlings and dirt in the feed. For the treatment of slurry, the capacity is reduced to about 1 ton per hour. The tables operate at 250 strokes per minute, with a length of stroke $\frac{7}{8}$ to 1 in. The refuse-discharge end is placed about $1\frac{1}{4}$ in. above the head end, or rather less for the treatment of fines. Transversely, from the feed to the coal-discharge side, the inclination is about $1\frac{3}{4}$ in. per ft. (or 1 in 7). About 35 gallons of water are required per minute. In operation, each table requires 1 h.p. for starting, but $\frac{3}{4}$ h.p. is sufficient when the table is running. Massco tables have not been used extensively.

The results in Table 90 were obtained in practice in New Mexico in cleaning coal for the Stag Canon Fuel Company. The raw coal contained about 25 per cent. of ash. The products were collected over distances of 1 ft. from the corner between the coal-discharge side and the head end, to the opposite corner between the refuse-discharge end and the feed side. Thus the quality of the products varies and, in general, becomes poorer in successive fractions.

TABLE 90.—RESULT OF WASHING MASSCO TABLE

No. of Fraction.	Weight of Fraction Gm. per min.	Ash Content per cent.	
1	376	7·0	53·0 per cent. of sample. 9·0 per cent. ash.
2	1,980	8·0	
3	2,952	7·6	
4	2,836	6·0	
5	6,000	7·0	
6	6,784	9·0	
7	7,446	10·8	
8	7,492	11·0	
9	3,464	11·0	11·3 per cent. of sample. 11·5 per cent. ash.
10	2,342	11·2	
11	1,906	13·0	
12	2,456	15·0	10·7 per cent. of sample. 22·0 per cent. ash.
13	2,824	23·0	
14	1,962	30·0	
		←— Corner of table.	
15	4,840	38·0	25·0 per cent. of sample. 52·7 per cent. ash.
16	3,922	47·0	
17	3,192	56·0	
18	2,662	68·0	
19	2,264	72·0	

It is unfortunate that these results do not show the float and sink figures for each fraction. They are, however, of especial interest because they clearly show one of the great advantages of table concentration, namely, that it is easy to control the exact quality of the cleaned coal by changing the position of the collector. No other coal-washing appliance is capable of such quick and easy adaptation.

Occasionally the standard Wilfley table is employed for coal washing, the deck being divided into two areas, a roughing (riffled) area and a finishing (smooth) area. The Wilfley deck is more commonly used when very fine coal is being treated (say, $\frac{1}{16}$ in. to 0), for in these circumstances the smallest particles can be effectively cleaned on the finishing area. With a completely riffled deck, a loss of very fine coal particles may be sustained in the refuse.

When the standard Wilfley table is in operation for coal washing, the raw coal is mixed with three times its weight of water, and, as it is essential that each particle of the feed be thoroughly wetted, it is usual to resort to wet grinding, when, as frequently occurs in America, the coal is crushed before washing.

Under proper working conditions, the coal and dirt separate into two distinct portions, there being a line of demarcation between the two (depending upon the amount of middlings). This line of

demarcation is plainly visible in many cases, but if the wet coal and wet shale are of similar colour and appearance it may not be quite clearly defined. The dividing line should coincide, in a general manner, with the diagonal line across the table along which the riffles terminate ; the coal should, as far as possible, be confined to the roughing surface (riffled) and the dirt to the finishing surface (smooth). The position of the line of demarcation can be varied by altering the side elevation of the table, and, in regular use, this is generally the only adjustment required to meet an irregular or a changed feed. By increasing the inclination, the line will move towards the coal-discharge point ; if the deck is more nearly horizontal it will be nearer the refuse-discharge end. The correct position is obtained when the line terminates at the lower corner of the refuse-discharge end.

The adjustments, other than the side elevation, that can be effected for any particular purpose are the end elevation, the amount of water mixed with the raw coal, and the length of stroke.

The Butchart Table.—The Butchart table was introduced in America in about 1916, and was said to be very efficient for coal washing, though it has never been widely used. The deck, which is covered with linoleum, is 15 ft. 6 in. by 6 ft., and is divided by the shape of the riffling into three areas, which may be seen in Fig. 125. In the middle area, the riffles are curved. The deck is of wood, bolted or riveted to steel plates, and is strengthened by additional steel plates along the sides and iron cross-pieces underneath. It is supported by four sliding bearings on the cross-pieces, which slide in receptacles on the understructure.

The whole table is mounted on two heavy steel base channels bolted to wooden or iron supports. The cast-iron pedestals, which provide the suspension bearings for the tilting beams, are bolted to the base channels. The base channels bend inwards at the head end of the table to support the driving mechanism.

The driving mechanism is of a modified toggle and spring type, enclosed in a cast-iron housing. The table receives an accelerated motion towards the refuse-discharge end, and a retarded motion in the reverse direction. The length of stroke ranges from $\frac{1}{2}$ to $1\frac{1}{4}$ in.

In order to obtain the necessary inclination of the table from the feed side to the coal-discharge side, the deck of the table is mounted, as stated, in bearings supported by cross-pieces on two tilting beams. The tilting mechanism consists of large-diameter bronze screws, operating in sleeves and actuated through bevel gearing by means of a handwheel. The gearing is so arranged that one turn of the handwheel increases the height of the feed side above the coal-discharge side by 1 in.

The riffling divides the deck into three portions. In the first and deepest portion, which extends from the head end to the curved portion of the riffles, the coal is cleaned roughly. The refuse sinks

to the bottom of the channels between the riffles and is moved towards the refuse-discharge end by the differential motion of the deck. Much of the cleanest coal is meanwhile washed over the riffles by the washing water and reaches the coal-discharge side. In the second portion, where the riffles are curved and bend towards the feed or upper side of the table, the refuse is rewashed. Here the washing water is deflected down the riffles and flows in a direction opposite to that taken by the refuse. The water flows down the slope towards the head end ; the refuse moves up the slope between the riffles as a result of the jerking of the deck. Consequently, any coal particles associated with the refuse on this portion of the table are washed backwards towards the roughing area.

The third portion of the deck is covered with straight riffles, parallel to the sides of the table. Its object is to discharge the refuse

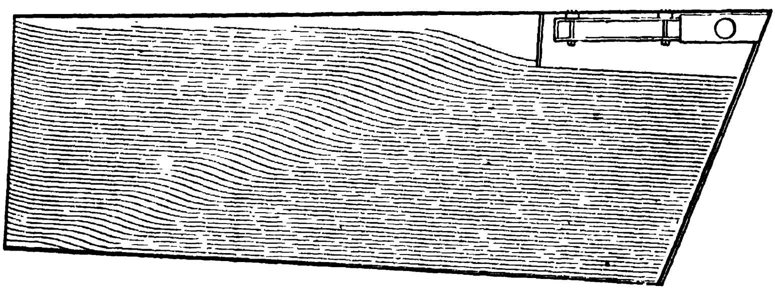

FIG. 125.—Deck of Butchart Table.

as near to the feed or upper side of the table as possible. By this means the middlings fraction is spread over a wide lateral area.

The following results of coal washing on a Butchart table were published by Prochaska ("Coal Washing," p. 223). They are the average figures obtained from a number of analyses made during the washing of an unsized Illinois coal from $\frac{3}{16}$ in. to 0 :—

	Raw Coal.	Cleaned Coal.	Refuse.
Ash per cent. . .	13·45	6·33	50·97
Sulphur per cent. . .	3·20	2·22	9·52

These results give no information as to the efficiency of washing, no reference being made to the loss of coal in the refuse or the amount of dirt in the washed coal. The following results (for which we are indebted to Mr. H. F. Yancey) relate to the washing of an Oklahoma coal, of size $\frac{1}{4}$ in. to 0, on a Butchart table :—

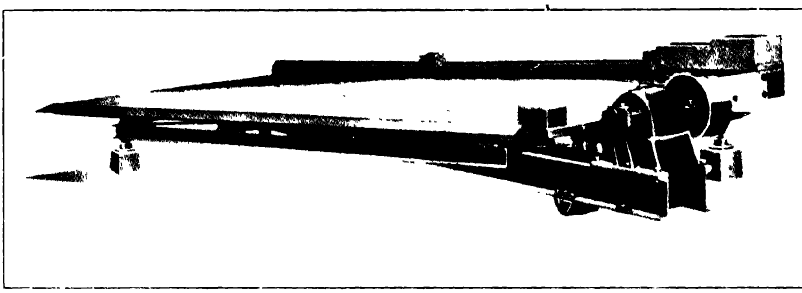

FIG. 126.—View of Deister Overstrom Table.

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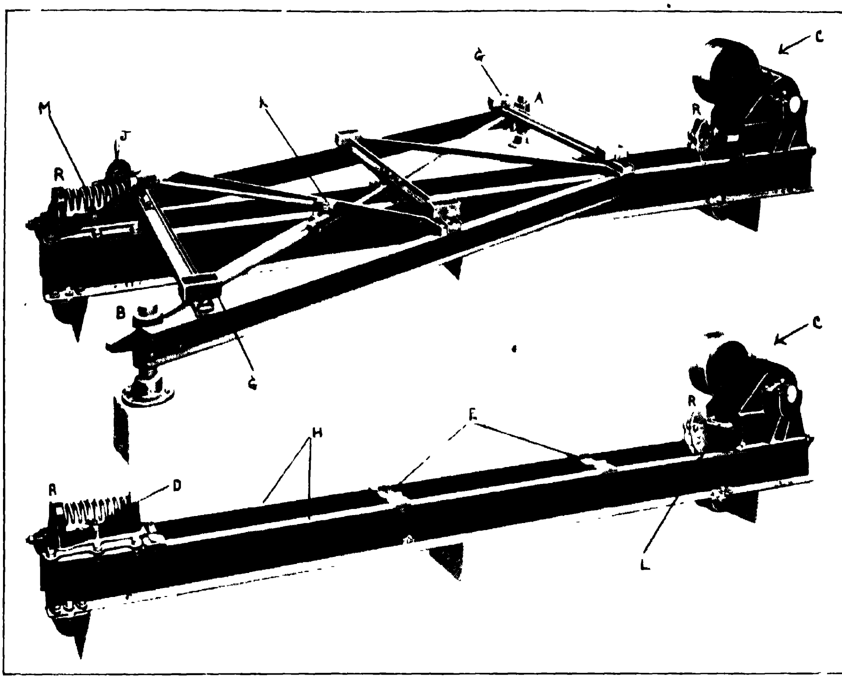

FIG. 127.—Understructure of Deister-Overstrom Table.

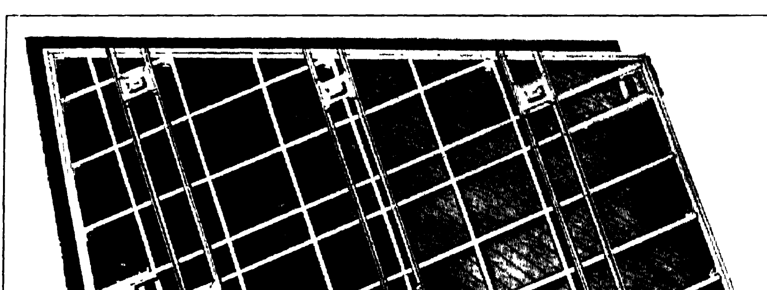

FIG. 128.—Underside of Deck of Deister-Overstrom Table.

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	Weight per cent.	Ash per cent.	Sulphur per cent.
Raw coal	100·0	29·6	1·18
Clean coal	68·2	10·2	0·96
Middlings	17·0	56·8	1·13
Refuse	14·8	76·6	1·84

The Deister-Overstrom Table.—The Deister-Overstrom diagonal deck coal-washing table was introduced in America before 1920, and the first large commercial unit was installed at the Wadesville Breaker of the Philadelphia and Reading Coal and Iron Company. The table was the outcome of the acquisition, by the Deister Concentrator Company, of the Overstrom patents, and the Deister-Overstrom table introduced features of each of the earlier types of table. It is said to be used for coal-washing in America in greater numbers than any other type of concentrating table.

A general view of the table is shown in Fig. 126, and the arrangement of the riffling on the deck of the table in Fig. 129. The table supports and the frame are shown in Fig. 127. The main base of the table consists of two 9 in. steel channels, H, riveted to bedplates which are attached to concrete foundation blocks. The head mechanism, C, is bolted to one end of the channels and the spring seat, D, to the other end. Two cradle seats, E, provide fixed supports for the tilting frame.

The deck of the table, the underside of which is shown in Fig. 128, is supported by six sliding bearings in the seatings, G. The seatings are fixed to the tilting frame, which consists of a braced steel framework, supported on the main base in cradle seats, and in operation they are filled with oil to give an easy sliding motion.

The tilting mechanism, which provides a slope down which the current of washing water will flow from the feed side to the coal-discharge side, consists of two slotted posts at the corners, A and B. The operation of the handwheel, J, actuates a rack and pinion, K, which moves two opposing wedges at A and B. The wedges move in slots in the tilting posts and raise or lower the corners of the framework by equal amounts, A being lowered if B is raised, or *vice versâ*. The variation in the inclination allows the deck to take up any position between the extremes—horizontal (from feed side to coal-discharge side), and with the uppermost corner of the feed side at a height of 2 ft. 10 in. above the opposite lowest corner. Under the latter conditions the inclination from corner to corner is 1 in 22, and along the end of the table 1 in 5.

The deck is kept in alignment by two rocker arms, R, one attached by means of a connecting link and steel yoke, L, to the head mechanism, and the other attached to the spring, M. Neither

rocker arm bears any of the weight of the table, and the sliding bearings under the deck play no part in preserving the alignment.

The deck itself is made of wooden strips fixed diagonally and liberally strengthened (Fig. 128). It is covered, on the upper side, with linoleum, on which the wooden riffles are laid. Its shape is different from that of the Wilfley (or Massco) and Butchart tables, being so designed that the material takes a diagonal course across the table. The head mechanism is fixed at one corner of the table and the motion is parallel to the riffles, but inclined to the feed and washed-coal sides. On the Wilfley deck, the riffles are parallel to the washed-coal discharge side. Instead of the refuse passing straight across the deck (as on the Wilfley), on the Deister-Overstrom table it travels diagonally from corner to corner of the deck. Its distance of travel for a given deck area is, therefore, greater, and the chances that coal particles will be lost in the refuse are consequently reduced. Furthermore, the riffles used are shallow and narrow, and the feed spreads out into a broad thin bed which greatly facilitates cleaning.

Figure 129 is a view of a Deister-Overstrom table in operation at Hazelton Shaft Colliery of the Lehigh Valley Coal Company, Pennsylvania. The raw coal, mixed with about twice its weight of water, is supplied from one corner of the table (right-hand side of Fig. 129) and water is delivered from the boxes running along the upper side. The riffles are arranged in groups. At intervals, one of the riffles is of greater height, near the head end, than the remaining riffles, and tapers down to approximately the same height as the other riffles near to the refuse-discharge end. This is the " Deister Pool " system.

The occasional higher riffle serves several purposes. The particles tend to bank up against the higher riffle, but coal particles are washed over it, whereas the middlings tend to accumulate in the bank and are moved by the head motion towards the refuse end. There, as the riffle decreases in height, they are separated from true refuse because they are able to pass over the riffle. It frequently happens that raw coal contains thin flat particles of dirt. On a concentrating table, these lamina frequently tend to be borne along on the top of the coal, floating, as it were, on coal particles, and are, therefore, difficult to remove. It is in such an event that the occasional high riffle serves a particularly useful purpose. As they progress across the table, floating on the coal, many of the flat dirt particles turn on to a side as they cascade over the high riffle, and are trapped below the coal. Once they are below the coal and between two riffles on the deck, they will be discharged into the refuse. On other riffling systems, with an even height of riffles, this would be less likely to occur.

The remaining riffles on the deck of the Deister-Overstrom table are of a uniform height of either $\frac{1}{8}$ in. or $\frac{1}{16}$ in., according to the size of coal being washed. They do not taper, as does the occasional

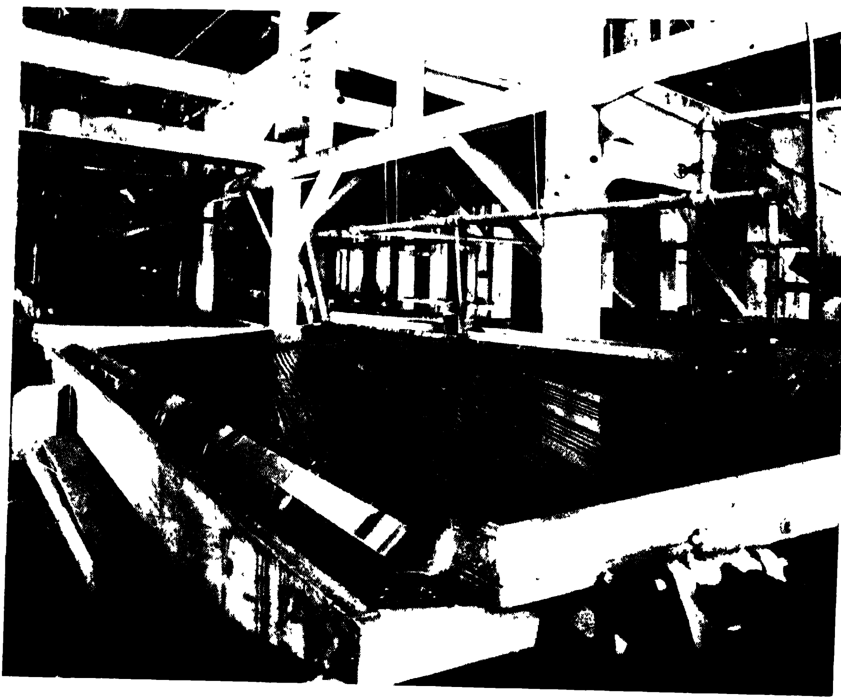

FIG. 120. Deister-Overstrom Table in Operation

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higher riffle. The riffles terminate on a diagonal line across the deck, which corresponds to the natural line of division between the clean coal and the refuse. The diagonal termination of the riffles extends for only about two-thirds of the length of the refuse-discharge end ; for the remaining one-third the riffles terminate at the end of the deck.

There are about 106 riffles on the standard Deister-Overstrom sand table used in ore-dressing practice ; on the No. 7 standard coal-washing table there are nearly 130 riffles. It will be recalled that on the Massco table there are 35 riffles ; on the Butchart there were about 50 riffles, and the total deck area of the three types of table is approximately the same. On the Deister-Overstrom table some of the riffles are quite short ; nevertheless the space between each pair of riffles is a washing area, and it is to be expected, therefore, that washing will be highly efficient. It is, indeed, claimed for the Deister-Overstrom table that, apart from an increased capacity, because of the diagonal travel of the refuse and the larger number of riffles, the cleaning effected compares favourably, in American practice, with that on any other type of coal-washing table.

Some portion of the efficiency of the table may be attributed to the small production of " middlings." It may here be remarked that the " middlings " produced in many coal-cleaning operations are not true middlings (that is, particles intermediate in density between the clean coal and the refuse) but consist of a mixture of coal and dirt, associated with only a small amount of truly middlings material. Such a " middlings " product consists largely of particles which have not gone where the designer of the plant expected them to go ; they are returned to the appliance in order that they may have another chance to go either with the clean coal or with the refuse. This occurs frequently in tabling processes (especially in dry tabling or pneumatic processes). This is prevented to a large extent on the Deister-Overstrom table because the deck is raised, forming a " high-spot " near the corner where the middlings tend to collect. When the coal is relatively free from middlings, the high-spot may cause a gap to form at the corner, but when the coal contains a large proportion of middlings, there is a gradual increase in density of the product round the discharge edges of the table. The portion between the clean coal and the refuse consists of true middlings, and can only be treated as a separate product for separate use, or for crushing and rewashing to recover the coal particles.

Since the Deister-Overstrom is a combination of two tables, there are available two types of head motion, the Deister motion and the Overstrom motion. Each of these is in common use on coal-washing tables. The Deister mechanism is the more common of the two and the principle of its action is similar to that described later for the Deister Plat-O table.

The results of a seventeen-hour test on the washing of pea size

($\frac{3}{4}$ to $\frac{3}{8}$ in.) anthracite were published by Griffin (*Coal Age*, August 14th, 1924). The table treated 20·2 tons of anthracite per hour, producing 13·2 tons of cleaned product. The results were as follows :—

Raw anthracite (pea) . . .	38·5 per cent. ash.
Clean anthracite . . .	5·74 per cent. ash.
Anthracite in refuse . . .	2·02 per cent.

Tests on smaller sizes of anthracite, all less than $\frac{3}{8}$ in., resulted as follows :—

Size.	No. 1 Buck-wheat.	Rice.	Barley.	No. 4 Buck-wheat.
Feed (tons per hour) . . .	19·27	17·95	14·43	2·0
Ash per cent. in feed . .	43·30	35·75	33·53	29·0
Ash per cent. in clean product .	15·43	15·03	17·46	9·5
Ash per cent. in refuse . .	80·60	78·19	74·72	—

The percentages of ash in the cleaned No. 1 buckwheat, the rice and the barley sizes are high. It should be noticed, however, that the ash contents of the refuse materials are very high. Moreover, the tables were being worked at their fullest capacity, and the chief aim would appear to have been to lose the minimum amount of coal. With ash contents of 80·6, 78·2 and 74·7 in the refuse, this was probably being accomplished.

The figures for the No. 4 buckwheat (which represent the average analyses for a month's operation) are remarkably good. No. 4 buckwheat is coal through a $\frac{3}{64}$ in. mesh sieve and remaining on a 200 (Tyler standard) mesh sieve. Treating this fine material, the ash content was reduced from 29 to 9·5 per cent. There may have been a considerable loss of coal in the refuse, but treating what is essentially a slime, this may not be an objection of serious moment if the ash content of the cleaned material can be so effectively reduced.

The Deister Plat-O Table.—The Plat-O coal-washing table is illustrated in Figs. 130 and 131.

The understructure is simple, consisting of three cross channels mounted on concrete piers. At each end of the channels is a housing containing a slipper bearing, in which the deck is supported. Three of these bearings form part of the tilting mechanism. A fourth concrete pier supports the main spring and the casting which forms the spring seat. With an underframe of longitudinal channels the spring and seat may be bolted to the main underframe.

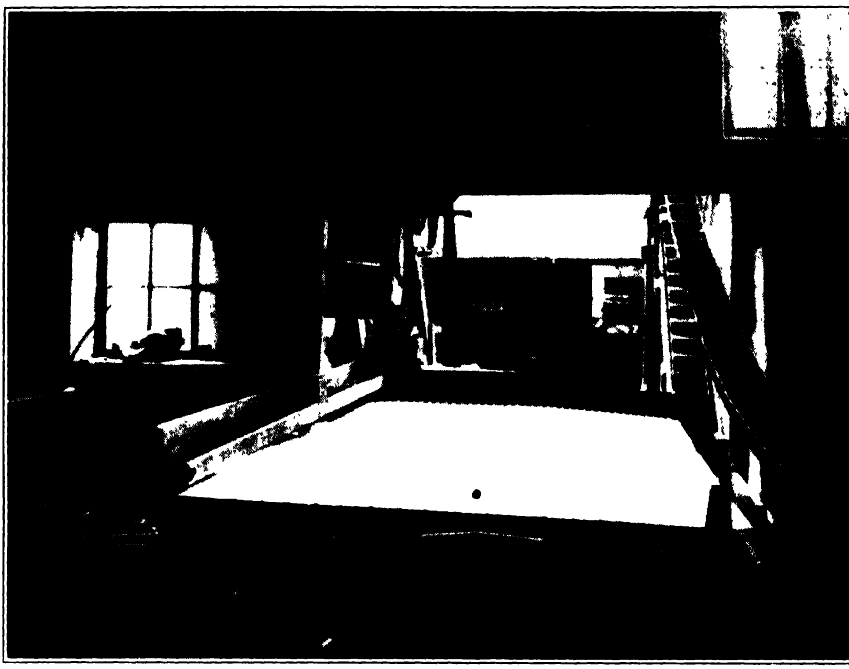

FIG. 130.—View of Deister Plat-O Table.

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The mechanism for tilting the deck consists of a rod to which three cast-iron wedges are locked in the proper positions. The wedges fit into the underside of the three slipper bearing housings, which are along one side of the framework. By the operation of a handwheel the wedges may be inserted further or withdrawn, to raise or lower the bearings along that side. The wedges are on the feed side of the table. In operation, the housings are filled with oil, and the deck of the table slides backwards and forwards in an oil bath.

The deck of the table is rectangular in plan and measures 14 ft. by 7 ft. It is made of diagonally laid wooden (cypress) strips, and is strengthened by stringers and head pieces made rigid by corner angles. On the under side are six bearing surfaces which fit and slide in the seatings on the frame. In section, the bearings form the arc of a circle, but alignment is maintained by a groove in the frame bearing and corresponding lip in the deck bearing. The deck is covered with linoleum (or rubber) to which wooden riffles are tacked. The rubber coverings employed are corrugated, the corrugations being about $\frac{1}{16}$ in. high. When linoleum is used, the covering is smooth.

The disposition of the riffles may be seen in Fig. 130, which is a view of a Plat-O table ready for operation, the near end being the refuse-discharge end. From the head or mechanism end, the riffles decrease in height towards the discharge end. Occasionally, especially for washing coal above $\frac{3}{8}$ in. in size, the riffles are made to terminate along a diagonal line as in ore-dressing on the Deister-Plat-O table, but in other cases they are continued, as shown, to the refuse-discharge end.

The Plat-O table has a raised portion near to the refuse end, to which its name (plateau) is due. After a particle of shale has travelled along the table (in Fig. 130 towards the observer) between two of the left-hand side, or feed side, riffles for about two-thirds of the length of the table, it is made to travel up a slight incline on to a plateau, a little (approximately $\frac{3}{8}$ to $\frac{1}{4}$ in.) above the level of its earlier path. Towards the right-hand side of the table the incline does not begin until the particle has travelled a greater distance, the incline commencing about 18 in. from the end of the table. When it reaches the incline, however, it rises to the same height above its earlier path as do particles on the left-hand or feed side. The incline thus extends diagonally across the table, and the inclined portion is 12 in. wide.

The table can therefore be regarded as being divided into two portions, the one near to the discharge end being slightly higher than the one near to the head end. The line of division between the two runs diagonally across the table and corresponds approximately to the natural line of separation between the coal and refuse (and hence to the dividing line between the roughing and finishing areas of the Wilfley table).

As the refuse passes up the incline, the washing water moves in an opposite direction down the incline, and so tends to remove any particles of coal entangled with the refuse. In this respect the incline corresponds to the curved portion of the riffling on the deck of the Butchart table.

The riffling of the deck of the Plat-O table may be seen from Fig. 130 to be of two kinds. Every second riffle of this table is a "pool" riffle, which extends right across the length of the table. Between the "pool" riffles are the main riffles, which, in the table illustrated in Fig. 130, terminate along a diagonal line across the table. This diagonal marks the top of the inclined portion of the deck. In the latest design of table, for washing coal of $\frac{3}{16}$ in. to 0 or $\frac{1}{4}$ in. to 0 size, the pool riffles, extending the full length of the table, are placed every third, instead of every second, riffle. All the riffles are $\frac{3}{8}$ in. wide, but vary in height. The "pool" riffles taper gradually from a height above the surface of $\frac{3}{8}$ in. at the head end to $\frac{1}{16}$ in. at the refuse end. The main riffles, two of which lie between each pair of "pool" riffles, are $\frac{7}{16}$ in. high at the head end and decrease in height to $\frac{3}{16}$ in. on a diagonal line across the top of the inclined portion of the deck. From this diagonal they are continued to the refuse end of the table by extension riffles, which decrease in height from $\frac{3}{16}$ to $\frac{1}{32}$ in. at the edge of the deck. Thus the table is riffled for the whole of its length.

For special purposes a different riffling system may be employed. For example, in one table designed for coal of size between $\frac{5}{8}$ and $\frac{3}{16}$ in., the riffles were inclined to the sides of the table for the greater part of their length, but on the plateau their direction changed and they became parallel to the sides. The system employed depends entirely upon the coal used. If the separation of coal from dirt is easy, the riffling can be simple. If it is more difficult, and coal tends to be discharged with the refuse, the riffles can be inclined towards the feed side, so that there is a greater tendency for the current to wash away light material from the refuse.

The feed to the Plat-O table is distributed from a square box at the corner between the head end and the feed side. The proportions of the coal and water in the feed recommended by the makers is about 1 to 1, but in operating practice it is found to be usually more satisfactory to employ a greater amount of water, the proportions being about 2 of water to 1 of coal. Such a proportion conforms more nearly to practice on concentrating tables of other makes. The wash water is supplied from two distributing boxes running along the feed side. The Plat-O table is designed for the treatment of coal of size less than $\frac{5}{8}$ in., and with a feed of coal of this size, the circulating water requirements are about 325 to 350 gallons per ton of coal.

The head motion is supported on a separate concrete pier, or, if main base channels are used, it is bolted to their ends. It is connected to the table by means of a steel yoke. A connecting rod

attached to the yoke engages in a seating in a steel plate bolted to the table, the actual height, relative to the deck of the table, at which connection is made being variable by providing several seatings. The yoke and connecting rod may be seen in Fig. 131, which is a drawing of the head mechanism of the table.

The motion of the deck is imparted by a toggle and lever mechanism acting with a spring. The differential motion is produced partly by this means and partly by the design of the eccentric

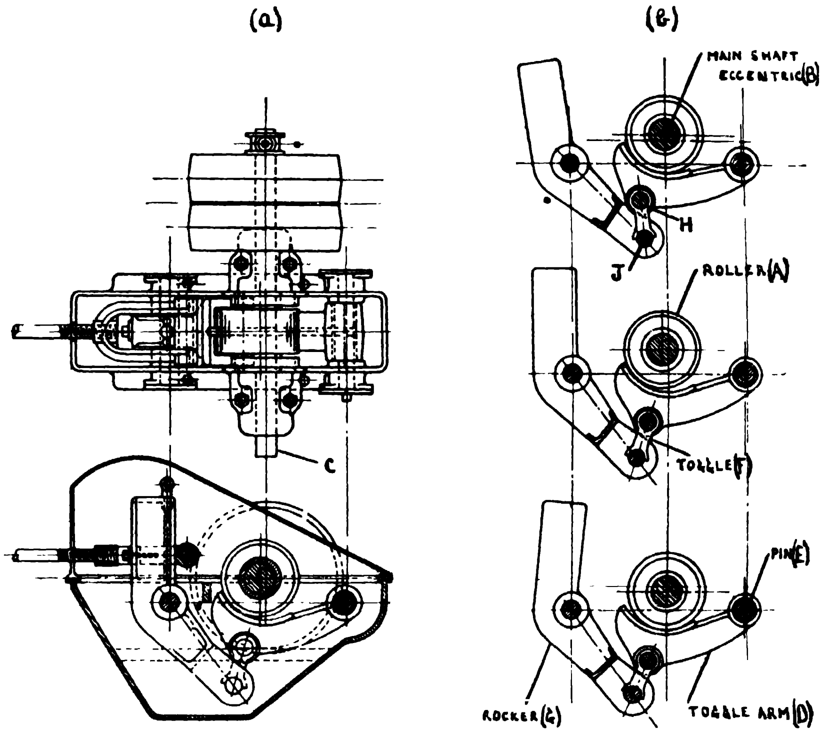

FIG. 131.—Head Mechanism : Deister Plat-O Table.

driving shaft, which imparts a varying angular speed to the roller actuating the toggle and lever mechanism.

The main shaft, C, has an eccentric portion, B, on which is mounted a loose roller, A. The shaft runs in two main bearings fixed to the frame of the table, and is driven by a 21 in. diameter pulley. The roller, A, bears against the surface of a steel toggle arm, D, which is pivoted at its other end by an eccentric pin, E. The motion of the toggle arm, D, is transferred by the short toggle, F, to the steel rocker arm, G, which is a bell-crank lever. The bearing upon which the rocker arm is pivoted, and the eccentric pin which pivots the toggle arm, are both rigid with the frame of the table.

The mechanism is completed by a spring mounted in a seating

near to the refuse end of the table. During the backward stroke it is compressed and is released during the forward stroke, so giving to the forward stroke an accelerated motion.

The table may be imagined to be at the end of its backward stroke and ready to move forward again. In this position the roller, toggle and rocker arm are in the positions shown in the lower diagram of Fig. 131 (*b*). The upper and lower bearings, H and J, of the toggle are then just out of alignment with the main shaft eccentric, the eccentric being slightly to the left.

The rotation of the eccentric shaft for a quarter of a revolution causes the roller to move forward slightly. When this occurs, the pressure of the spring, which was compressed during the backward stroke, causes the upper toggle joint, H, to move forward, thereby causing the toggle arm, D, to rise slightly to the position shown in the middle diagram. As the eccentric shaft rotates through another 90 degrees, the upper toggle joint moves further forward and the toggle arm is raised still further. The raising of the toggle arm allows the lower end of the rocking arm to rise and causes the upper end of the lever to move forward. The motion imparted to the upper arm of the rocker, with the further assistance of the spring, causes the forward motion to be accelerated. The sudden reversal of the movement, and the retarded backward motion, have the effect of jerking the refuse forward between the riffles.

The mechanism of the Deister Plat-O table is capable of several adjustments. It will be seen in Fig. 131 (*a*), that the yoke and connecting rod are not fixed to the end of the rocker arm. By varying the position of the yoke on the rocker arm the length of the stroke can be altered. If the distance between the connection and the centre pivotal bearing of the rocker arm is increased, the length of stroke is increased, and *vice versâ*. The adjustment is made by a screw and handwheel.

The differential motion can be altered by changing the size of the roller, A, the length of the toggle, F, and the position of the toggle arm, D. The latter adjustment is made possible by the use of an eccentric pin, E, as the pivotal bearing. If the eccentric portion of the pin is towards the table, the toggle arm is moved forward, and the position of the upper toggle joint, H, relative to the lower joint, J, is changed in such a manner that the sharpness of the "kick" of the motion is increased. If the eccentric pin is reversed, the joint, H, moves backwards relative to J and a smoother reciprocating movement results.

The Plat-O head motion is totally enclosed in an iron casing which is half-filled with oil. The motion is self-oiling by splashing. As with other tables, the horse-power required is about 1 h.p. for starting and about $\frac{3}{4}$ h.p. for running.

The capacity of the table is stated to be between 7 and 10 tons per hour of feed coal of size $\frac{1}{4}$ or $\frac{3}{8}$ in. to 0. The capacity is, of course, lower the greater the amount of refuse and the smaller the

size of the coal. For normal operation on coal of this size the length of stroke is about $\frac{7}{8}$ in., and about 285 strokes are made per minute.

A certain number of Deister Plat-O tables have been erected in this country for coal washing. At one colliery in South Yorkshire a table was installed for the treatment of crushed washery refuse. Under one test the table had a capacity approaching 10 tons per hour, and a good separation of coal and dirt appeared to be effected. In subsequent operation, however, the table proved unsatisfactory, both as regards efficiency and capacity. Unfortunately the tests were abandoned before they could be considered to have been prosecuted exhaustively.

The results in Table 91 were obtained by Mr. B. M. Bird in a test at a washery in Pierce County, Washington, on the washing of middlings crushed to pass a $\frac{3}{8}$ in. screen. The table used was a standard Deister Plat-O table, but the deck had been modified, the plateau at the middlings corner being cut away prior to the test.

TABLE 91.—FLOAT AND SINK TEST ON RAW COAL

S.G.	Weight of Sample per cent.	Ash per cent.	Weight cumulative.	Ash per cent cumulative.
< 1·38 .	63·9	7·3	63·9	7·3
1·38 to 1·50 .	12·7	24·0	76·6	10·1
1·50 to 1·70 .	12·5	38·3	89·1	14·0
> 1·70 .	10·9	62·8	100·0	19·3

WASHING RESULTS ON PLAT-O TABLE

Zones, 1 ft. Wide.	Weight of Sample per cent.	Ash per cent.	Weight cumulative.	Ash Per cent. cumulative.
1 and 2	3·3	8·7	3·3	8·7
3 „ 4	4·5	7·6	7·8	8·1
5 „ 6	13·7	9·6	21·5	9·0
7 „ 8	13·2	11·1	34·7	9·8
9 „ 10	15·6	12·1	50·3	10·5
11 „ 12	14·2	13·3	64·5	11·1
13 „ 14	12·0	21·5	76·5	12·8
15 „ 16	6·6	36·4	83·1	14·6
17 „ 18	6·5	38·4	89·6	16·4
19 „ 20	7·0	46·8	96·6	18·6
21	3·4	51·4	100·0	19·7

These results indicate the possibility of the recovery of coal from the middlings made by another process of washing. Whereas the bulk sample contained nearly 20 per cent. of ash, 76 per cent. can be recovered as coal if an ash content of 12·8 per cent. is permissible.

The H.H. Concentrating Table.—The H.H. table (Overstrom Universal), made by Messrs. Huntingdon Heberlein & Co., Ltd., is the only table which has been utilised to any great extent for coal washing in this country. It differs chiefly from the tables which have been described in the method of support and in the driving mechanism. It is shown in Fig. 132.

The understructure (Fig. 133) is similar to that first used on the old Ferraris or Buss table, erected on the Continent. Instead of a rigid structure upon which the deck of the table moves backwards and forwards on some form of sliding or roller bearings, the deck is supported on flexible wooden legs. On the H.H. table, ten laminated legs, composed of strips of hickory wood or ash, form the supports for the framework of the deck. The legs are greased and wrapped with tape to preserve them. The wooden or steel floor beams, to which the legs are bracketed with steel angle-pieces, are braced by two cross-pieces and by a bumping block and post.

The tops of the legs are bracketed to the shaking frame on which the deck itself rests. The shaking frame consists of longitudinal wooden or steel stringers adequately strengthened. In the latest models the shaking frame is a wooden flooring to which the deck is rigidly attached. The actuating mechanism is fixed to the shaking frame.

On the Overstrom Universal table the deck itself was hinged between brackets attached to the shaking frame along the tailings side, but along the feed side it rested on wedge supporting pieces. The wedges were movable longitudinally by the operation of a hand-wheel, and by this means the feed side of the table could be raised to give the required transverse inclination. According to Grounds (*Proc.* S. Wales Inst. of Eng., 1927, 42, 545) the practice of equipping the H.H. table with an adjustable transverse inclination has been abandoned, and the tables supplied have a fixed slope, which varies according to the size of the coal. The inclination provided is 1 in 8·35 for $\frac{1}{4}$ in. to 0 coal; 1 in 7·11 for $\frac{5}{8}$ to $\frac{1}{4}$ in. coal; 1 in 5·49 for 1 to $\frac{5}{8}$ in. coal. At the refuse or concentrates end, and along the feed side, the deck is held rigidly in a locking device which prevents longitudinal movement between the deck and the frame.

The ten wooden legs of the H.H. table, which act as springs, are inclined slightly towards the head mechanism. The consequence is that, as the table is thrown forward, the deck describes the arc of a circle, rising slightly. This slight inclination, besides tending to cause the washing water to flow in a direction opposite to that of the refuse, results in the agitation and better stratification of the bed.

A complete H.H. table for washing small coal is shown in Fig. 132,

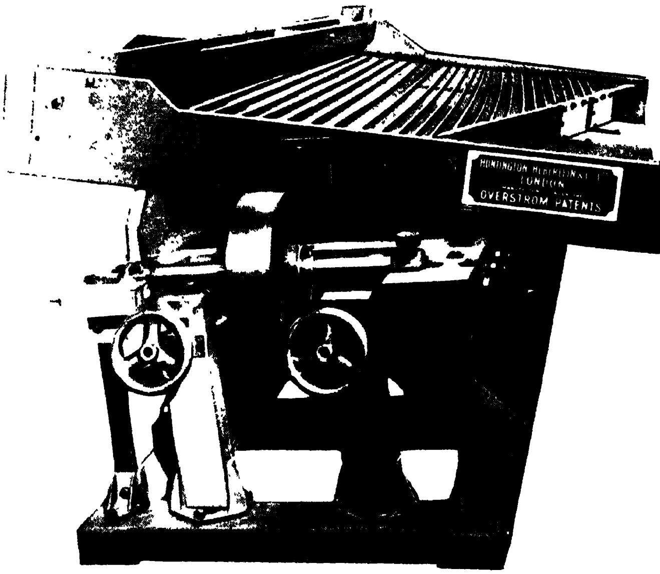

Fig. 132.—View of H. H. Table.

Fig. 133.—Understructure of H. H. Table.

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in which the curved shape of the riffling may be seen. The object of the curve is that the refuse, before passing to the discharge collector, passes up the slope from the feed to the coal-discharge side, and therefore moves against the water current. The coal entangled with the refuse is washed backwards by the water current, the refuse passing forward between the riffles because of the impetus given to it by the motion of the deck. The effect is similar to that produced in the curved portion of the deck of the Butchart table, and the raised portion of the Deister Plat-O deck. For the larger sizes of coal, straight riffles may frequently be employed. There is often a slight tendency for the refuse to bank where its path bends upwards, and this is more especially the case with larger lumps. At Weetslade Colliery, Northumberland, where four H.H. tables have been in use for some time, the coal is sized between the limits $\frac{7}{8}$ to $\frac{5}{8}$ in., $\frac{5}{8}$ to $\frac{3}{8}$ in., $\frac{3}{8}$ to $\frac{1}{8}$ in. Curved riffles are used for the smaller sizes, and straight riffles for the larger size.

For all sizes of coal, the height of the riffles decreases towards the refuse-discharge end to allow middlings particles to pass over the riffles and find their way towards the corner of the coal-discharge side. Their height and the distance apart vary according to circumstances. At Weetslade, one of the tables has been fitted with a rubber surface, and the riffles themselves are rubber faced on the side facing the feed, to prevent excessive wear. The remaining tables are covered with linoleum and fitted with soft wood riffles. For coal washing, a rectangular deck 15 ft. by 7 ft. is employed.

The coal is fed to the table from a box near to the mechanism end, and washing water is supplied from a box running along the feed side. The feed consists of coal with one and a half times to twice its weight of water; circulating water is fed along the greater part of the table, and make-up water is supplied near to the refuse-discharge end.

The actuating mechanism of the H.H. table is very simple. It consists of an unbalanced loose pulley, driven on a shaft fixed rigidly across the shaking frame. As the unbalanced weight revolves, its centrifugal force imparts a reciprocating horizontal motion to the frame. The pulley is mounted between collars, on a shaft, which is clamped to a heavy steel (or wooden) yoke. At its other end, the yoke is fixed rigidly to the head end of the table framework. Two springs, one of which is just visible in Fig. 132, press at one end against the cross piece of the yoke. At the other end, the springs press against the top of posts hinged in castings in the floor. The springs thus accelerate the forward stroke of the motion, but act as a cushion and retard the backward stroke.

The forward stroke is terminated by a bumping block supported on the framework of the table, but rigidly fixed to the floor of the building. The bump occurs between the post and the cross piece of the yoke. To soften the bump, the face of the yoke is padded with a cushion made of strips of canvas. Further padding is provided by

pieces of canvas hanging from the head piece of the table and interposed between the bumping post and the yoke.

The motion therefore consists of a forward throw by the loose pulley which is accelerated by the springs and is terminated with a bump by contact with the rigid bumping beam. The shock of the bump is absorbed to a certain extent by the pads of canvas, but the stroke is reversed sufficiently rapidly to cause the heavy material settled between riffles to be jerked along towards the discharge end of the table. When the stroke is reversed, the motion is retarded by the springs, thereby aiding a fairly smooth return and reversal to a second forward throw.

The tension of the springs is varied by means of the handwheels shown in Fig. 132. The length of the stroke is altered by varying the tension of the springs, and also by removing one or more of the canvas strips hanging from the table and inserted between the pad on the yoke and the bumping post. The length of stroke usually employed is $\frac{1}{2}$ to $\frac{3}{4}$ in. A longer stroke is required for larger coal or for coal containing a considerable amount of refuse. The weighted pulley is revolved at about 240 r.p.m., or at slightly higher speeds for smaller coal.

One advantage claimed for the H.H. table is that, because of the inclination of the spring legs towards the head end, and the consequent upward throw of the table during the forward stroke, the length of the stroke imparted by a loose weight revolving at constant speed is automatically regulated to a certain extent with changes in feed. With a heavier feed, the stroke will lengthen and will shorten with a lighter feed.

The Overstrom Universal table was used to a considerable extent for coal washing in America. It has not, however, received the favour granted to other makes of table. This is no doubt due in part to a prejudice against bumping tables. In comparison with jerking tables, with head motions of the Wilfley or Deister type, bumping tables have been found in ore-dressing practice to be harmful to the structure of the building in which they are housed, and generally to be expensive in upkeep, because of the wear and tear involved. Moreover, the bumping action is found occasionally to interfere with good stratification of the bed.

In the H.H. table, these objections are overcome by means of the canvas pad between the table and the bumping post, and the cushion springs, and by rigid supports. The shock experienced is then little more than with a jigging screen.

The capacity of the H.H. table with different sizes of coal is as follows :—

Size (in.)	Capacity tons per hr.)
$\frac{5}{8}$ to $\frac{3}{8}$	15
$\frac{3}{8}$,, $\frac{1}{8}$	10
$\frac{1}{8}$,, $\frac{1}{16}$	5

The washery at Weetslade Colliery, Northumberland, is shown in section in Fig. 134. The raw coal is delivered from the colliery screening plant into the receiving hopper, which has a capacity of 20 tons. It is delivered into the buckets of the raw-coal elevator by a jigging feeder and elevated to a storage bunker at the top of the washery building. From the bunker, it falls on to vibrating

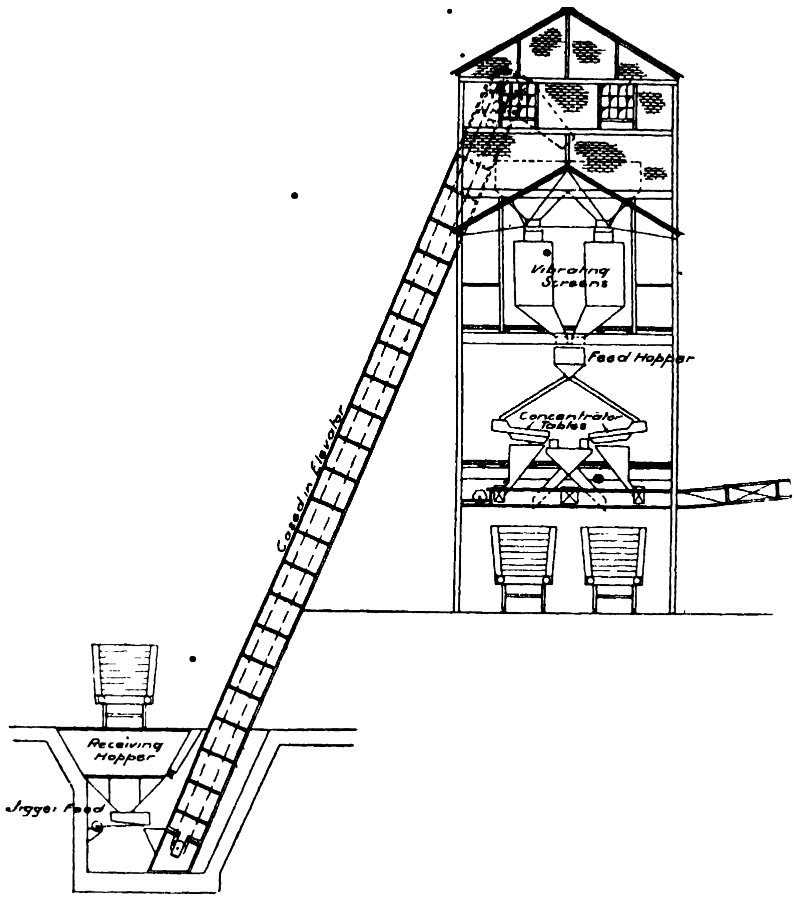

FIG. 134.—Cross-section through Washery using H.H. Tables.

screens which divide it into four sizes, namely, $\frac{7}{8}$ to $\frac{5}{8}$ in., $\frac{5}{8}$ to $\frac{3}{8}$ in., $\frac{3}{8}$ to $\frac{1}{8}$ in. and $\frac{1}{8}$ in. to 0. Two sets of screens of the H. H. Overstrom vibrating type are installed, each with a capacity of 50 tons per hour. The sized fractions are fed to feed hoppers placed adjacent to the tables.

The two largest sizes of coal are treated on separate tables, the $\frac{3}{8}$ to $\frac{1}{8}$ in. fraction being divided and washed on two other tables. The $\frac{1}{8}$ in. to 0 fraction is by-passed and is not washed. The washed

coal passes from the concentrating tables over dewatering screens, attached to the delivery ends of the tables, on to a conveyor, which collects and combines all three sizes of washed coal. The unwashed $\frac{1}{8}$ in. to 0 fraction is mixed with the washed coal on the conveyor. The refuse from the tables passes down shoots to two dirt conveyors, shown extending from the right of the building in Fig. 134.

The plant, with a total capacity of about 40 tons per hour, is driven by one 50 h.p. motor, separate motors being provided for the raw-coal elevator and the water pump. The pump motor is 20 h.p., and the pump is capable of delivering 500 gallons per minute against a head of 65 ft.

Results of washing on H.H. tables, given by Grounds (*loc. cit.*) are reproduced in Table 92. The coal was from Lancashire.

TABLE 92.—RESULTS OF WASHING. H.H. TABLE

S.G.	Raw Coal.		Clean Coal.		Middlings.		Refuse.	
	Per cent of Sample.	Ash per cent.	Per cent of Sample	Ash per cent.	Per cent of Sample.	Ash per cent.	Per cent. of Sample	Ash per cent.
< 1·4	55·3	3·3	96·9	2·5	5·7	10·0	0·1	5·0
1·4 to 1·6	4·4	18·0	2·5	23·0	15·4	29·9	0·15	22·0
> 1·6	40·3	82·1	0·6	42·0	78·9	61·7	99·75	84·3
Total	100·0	35·7	100·0	3·2	100·0	53·8	100·0	84·1

These results of washing are very good, the clean coal contains very little true refuse, and the refuse practically no coal. The raw coal, however, is an easy one to wash, for although it contains 35·7 per cent. of ash, the ash is concentrated in a very heavy fraction. In this test, nearly 6 per cent. of the coal was collected as a separate middlings fraction containing 53·8 per cent. of ash. The coal treated was $\frac{1}{8}$ in. to 0.

An H.H. table has been in operation at Barnsley Main Colliery for over two years washing the slurry made in a Baum washery. The slurry is partially dewatered on a $\frac{1}{2}$ mm. screen and is fed to the table at a rate of about 5 tons per hour. The heaviest dirt particles are removed and the washed slurry is collected and mixed with the coking slack. From the 5 tons of slurry fed per hour, 18 cwt. of refuse are removed with an ash content of 50 per cent. The refuse from the table passes by a shoot directly to the boot of the dirt elevator in the Baum washery. The raw slurry is delivered to the table by a slurry pump and passes from the dewatering screen into

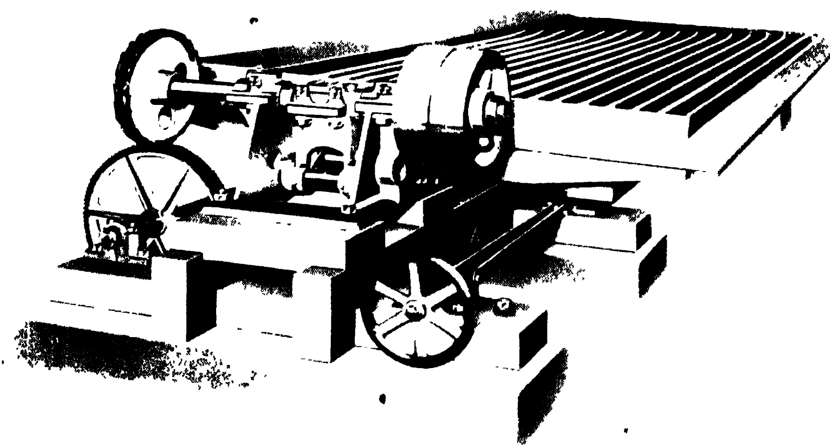

FIG. 135. View of the Broadway Ta

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the table feed-box. The water supplied to the table is the make-up water for the Baum washery (about $7\frac{1}{2}$ tons per hour).

The table is a standard table with a slightly modified system of riffling, and it gives satisfactory results even when fed at four times the load for which it was designed (1 ton per hr.).

The Broadway Table.—The Broadway table, made by Messrs. Bowes-Scott and Western, Ltd., is an adaptation of their well-known ore-dressing table for the purpose of washing small coal. The table is shown in Fig. 135 and the driving mechanism in Fig. 136.

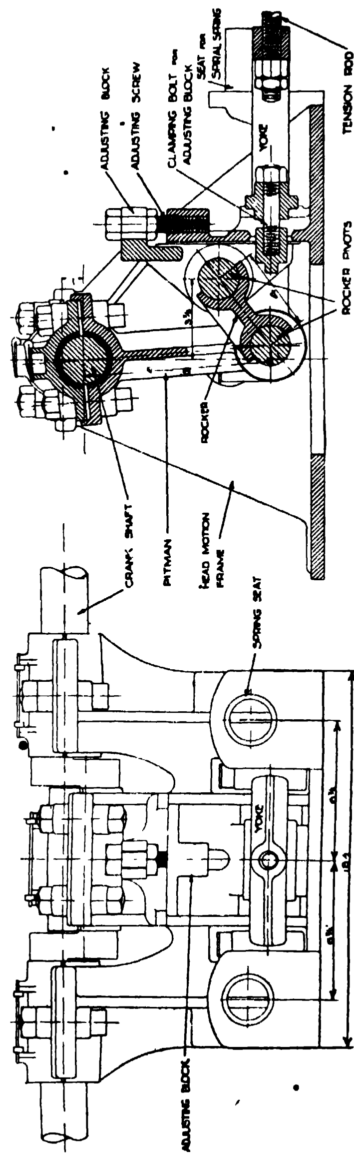

Fig. 136.—The Broadway Table Head Mechanism.

The deck is 16 ft. long by 6 ft. wide, and is covered with linoleum to which teak riffles are attached. The principal feature of the table is a lateral rocking movement in addition to the customary reciprocating longitudinal motion. The rocking assists the light coal particles to pass over the riffles without disturbing the layers of dirt below. By this means, the capacity of the table is increased.

The driving mechanism consists of a cam-shaft, and the motion is transmitted to the table by a steel yoke, one end of which is fastened by a hinge-pin bracket to the table, the other end being attached to the headgear. The forward stroke is accelerated by a pair of strong springs, which cause a rapid reversal and a retarded backward motion. About 250 strokes are made per minute. The rocking motion is produced by a longitudinal cam-shaft, driven by an enclosed spiral-worm gear from the main-shaft.

The deck is supported on rollers working in guides on cross pieces; the reciprocating motion moves the deck in the guides and

the lateral motion is applied to the cross pieces, which act as a rocking frame. The understructure consists of three concrete blocks for the table and one for the driving mechanism.

The table erected at the Garswood Hall Colliery, Lancashire, has been in operation for nearly a year. The coal treated is fines through a $\frac{3}{32}$ in. screen with which a capacity of about 4 tons per hour can be attained. Before being fed to the table, the coal is fed into a cylindrical vessel in which it is mixed with water by a revolving paddle. This ensures proper wetting of the particles. The results of washing are as follows:—

	Ash content per cent.
Raw coal	29·5
Washed coal	9·4
Refuse	78·7

Another Broadway table installed at Plean Colliery is used for rewashing a portion of the dirt from a Lührig jig washer. On account of segregation, the material delivered from the buckets of the dirt elevator is richer in coal at one side than at the other. The richer portion is passed by a shoot to a Broadway table. The float and sink analysis of the feed and of the products is given in Table 93.

TABLE 93.—RESULTS OF TREATING JIG-WASHERY REFUSE ON BROADWAY TABLE. AVERAGE OF TWO TESTS

Material.	Weight % of Total.	% Ash.	S.G. < 1·4.	S.G. 1·4—1·6.	S.G. > 1·6.
Feed	100·0	49·1	35·4	3·8	60·8
Clean coal	25·0	5·1	93·9	4·6	1·5
Middlings	15·4	25·4	53·6	12·2	34·2
Refuse	59·6	71·2	2·4	2·2	95·4

Miscellaneous Concentrating Tables.—Among other tables which are in use for ore-dressing are the James table and the Record table. The James table has not, so far as we know, been used for coal-washing, but its design is interesting because, in some forms, the deck consists of different planes, upon which different degrees of separation are effected, and the riffling and differential motion are, as on the Deister-Overstrom table, inclined at an angle of 30 degrees to the sides. It is used with a small stroke (about $\frac{1}{2}$ in.) and a rapid rate of vibration (about 300 per minute). The James sand table is riffled with brass strips $\frac{1}{16}$ in. high and $\frac{3}{8}$ in. wide, whereas the slime table is unriffled.

The Record table, which also, so far as we know, has not been

used for coal-washing, has an ingenious head mechanism. The deck is in many respects similar to that of the Wilfley table, but the understructure is unusual, consisting of a wooden framework topped by two horizontal beams upon which the deck slides to and fro. A diagram of the differential motion is given in Fig. 137. The driving crank revolves about a shaft at regular speed, its various positions being shown by the numbers 1 to 8. The motion is conveyed to the following crank by a drag link, and the resulting irregular angular movement is represented by the corresponding positions 1 to 8 shown. By this means the shaft upon which the following crank revolves is driven at an irregular speed. The table is driven by an eccentric attached to this shaft and, instead of a purely harmonic motion as the eccentric alone would produce, a differential motion results. The distance between the two shaft centres is 3½ in., and the drag link is a little longer than this.

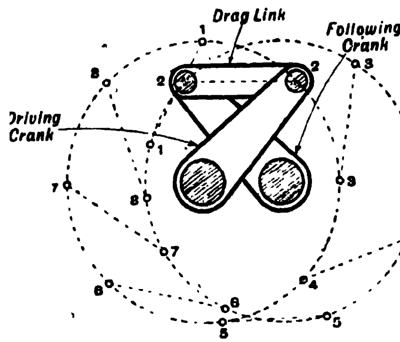

FIG. 137.—Diagram of Mechanism: Record Table.

The length of stroke of the Record table is about 3½ in. and the rate about 100 per minute. In practice it is said that classification of fine material is inaccurate, and that it is, therefore, only satisfactory as a roughing table.

GENERAL CONSIDERATIONS

Concentrating tables are used universally in ore-dressing, chiefly with material too fine to be treated in jigs, or for the further enrichment of jig products, and especially for the recovery of "values" from the middlings product of a jig, either after crushing or without crushing. Their use is justified by the value of the ore. Probably also, in cleaning a coal containing large quantities of middlings, the collection of a middlings fraction and its re-cleaning on tables after crushing would be a more profitable operation than is usually considered to be the case. In America, where in certain States it is necessary to mine dirty seams, and where washing is a necessary practice (*e.g.*, in Washington and Alabama), it is quite common to find concentrating tables as accessories to jigs for the treatment of uncrushed middlings.

In England, the applicability of table concentration is probably not yet fully appreciated, largely because many of the plants which have been installed can only be regarded as experimental plants. They have been, as a rule, supplied by the makers of ore-dressing

appliances, who, themselves, have had little direct experience of coal cleaning. In many cases the use of tables has proved entirely satisfactory, and there is little doubt that, for the treatment of fines, they are an important alternative to other methods.

It will be observed that all the tables described are said to be able to treat efficiently unsized material from, say, ¼ in. to 0. This claim can be justified by results provided that the material consists largely of pure coal and pure dirt particles. If there is a considerable amount of material intermediate in density between the two, then the cleaning is not always satisfactory, because coal and shale are both trapped among the middlings product and are carried in a given direction on the table much further than would otherwise be the case. If this middlings product is only present in small quantities, a good separation can be effected ; the recovery of coal is high and the heavy refuse is almost completely removed. In America, tables are chiefly employed in those coalfields where the coal contains relatively large quantities of an impurity known as "bone" coal. "Bone" is a material of fairly uniform composition and high ash content ; its specific gravity falling between 1·5 and 1·8. In England, coal between S.G. 1·5 and 1·8 almost invariably consists of interstratified particles of coal and shale, which can be separated by crushing. American bone-coal, however, is uniform, and its quality cannot be improved by crushing. If the fines contain large quantities of bone-coal, no cleaning process can effect a considerable reduction in the ash content without a considerable loss of combustible matter.

Bird ("Bureau of Mines," Rep. of Investigations, No. 2755, 1926) in a study of the sizing action of a coal-washing table shows the difficulty of treating an unclassified feed with a high content of bone-coal. He found that particles of coal (S.G. < 1·38) could just be separated from light bone (S.G. 1·38 to 1·50) if the ratio of the diameters of the particles was 4 to 3. Similarly, a sizing ratio of about 3 to 1 would be required if the coal were to be separated completely from the dirt (S.G. > 1·70). He suggests that, before treatment on a concentrating table, raw coal could advantageously be classified in a hindered-settling classifier.

In a later paper (" [illegible], 31, 670), Bird describes experiments using modified systems of riffling by which the efficiency of washing a coal with a high content of material of S.G. 1·38 to 1·70 is considerably increased. The modified riffling, which consists largely of introducing a larger number of riffles of greater height than the average, and of placing them closer together than is usual, enables a wider range of sizes to be treated simultaneously. It is evident, however, that with a coal fairly easy to wash, and under the most favourable conditions of operation, some degree of preliminary sizing of the raw coal is required if a high efficiency is to be attained. Probably a ratio of about 6 to 1 is the maximum, and this ratio will be reduced if the coal is rich in middlings.

In ore-dressing practice tables are used for treating the finest slimes, down to 200 mesh size. The results quoted for the Deister-Overstrom table, washing anthracite from $\frac{3}{64}$ in. to 200 mesh, in which the ash content was reduced from 29 per cent. to 9·5 per cent., amply justify the claim that tables offer a promising means of cleaning many examples of slurry, particularly if a special slime table be employed. Consequently, it is reasonable to suggest that a complete washery might be equipped with some suitable plant for cleaning large coal and concentrating tables for cleaning the fines, and that every size of the coal would, therefore, be efficiently cleaned. There are advantages in such a scheme in that tables require little horse-power, the supporting structure can be made lighter than for jigs, and the washery building need not be so massive. With modern smooth-acting tables there is little wear and tear, either to the machinery or to the building, and the capacity for a given floor space is high.

A great advantage of tabling is that the process is visible to the operator. Not only can he often tell by inspection when an adjustment is required, but he can see immediately the qualitative effect of that adjustment. On the other hand, in a large plant of, say, 150 tons per hour capacity, the individual units are distributed over a wide area. For this reason, more supervision is probably necessary than in a jig washer of the Baum type which can efficiently treat large quantities of unsized coal in one or two boxes. Concentrating tables do not deal efficiently with an irregular and intermittent feed. If the amount of dirt falls, coal moves further across the table than usual, and if the amount of dirt is restored to the normal, coal is lost in the refuse. With an increased proportion of dirt, the spaces between the riffles is filled and the washed coal contains particles of dirt. When the feed is irregular, especially if the feed is periodically interrupted, the whole load moves forward across the table towards the refuse-discharge point, and the working is most unsatisfactory.

Deck Covering and Riffling.—Nearly all the tables described are covered with a sheet of $\frac{1}{8}$ in. linoleum, to which wooden riffles are tacked. Linoleum is durable, light, and impervious to water. It seems to exercise just the right hold on the particles and to be in every way satisfactory. Usually a linoleum covering lasts for two or three years without replacement, but the riffles wear and require renewal every few months. Thicker linoleum ($\frac{1}{4}$ in.) has been tried, but its greater expense and the lack of suppleness in laying, make it unprofitable. Rubber covering for the deck has been tried, and although it is used successfully in certain instances, it is not accepted in general practice as a satisfactory substitute for linoleum. The particles have a tendency to stick to the rubber and the refuse (in ore-dressing, the concentrates) are not properly and freely discharged; moreover, if, by accident, a little oil is spilled on the rubber surface,

the rubber swells and forms a bump, which can be very harmful to efficient washing, and which cannot be removed by any known means. Glass surfaces have been used, with glass riffles or ridges incorporated in the surface. Concrete tops have also been employed, but they are heavy and require wooden nailing strips as seats for the riffles.

The riffles laid on the surface of concentrating tables are square-sectioned strips of wood nailed to the deck. In certain ore-dressing processes, where shallow riffles are required, strips of brass are frequently used, but they are expensive. Wooden riffles last, usually, for some months, and a table can be relaid in a few hours at little cost. As a rule soft pine wood is employed for riffles, but for coarse feeds, and therefore frequently for coal washing, oak is more satisfactory. Hard wood, however, has a tendency to chip at the edges. Softer wood, being more resilient, does not suffer from this defect. For certain operations, especially when treating very fine slimes, a preference is sometimes shown for grooves sunk into the surface of the table instead of riffles raised above it. This is the chief feature of the design of the Card table. The grooves, instead of being square-sectioned, have a flat angular section with a gentle slope downwards in the direction of the flow of the water current and a slightly steeper incline against the water current. They are deep near to the head end of the table, and taper until they vanish towards the concentrates end. On a grooved deck the eddy currents, which always arise on a riffled deck, are eliminated, and, for this reason, a grooved deck would appear to be particularly suitable for slurry washing.

Slimes.—When a table is set to treat an unclassified feed below, say, $\frac{1}{4}$ in., difficulty is sometimes experienced in that the finest particles tend to form a slime which, instead of stratifying, runs across the head side of the table and is delivered at the first portion of the coal discharge (or tailings) side. The particles which contribute to this slime are smaller than the average particles in washery slurry, being usually small enough to pass through a 200 mesh sieve, and they are frequently very dirty. Their actual constitution depends essentially upon the nature of the coal. They may, for example, be the product of disintegration of the shale when it comes into contact with the washing water. On the other hand, if the coal has been crushed, they may consist almost entirely of coal. Obviously, if these slimes are very dirty, they must be kept apart from the cleaned coal by using a suitable de-watering screen.

The effect of the presence of slimes in the washed coal is plainly shown in the following results published by McMillan and Bird (*loc. cit.*)

In the second example there were a further 10 subsequent zones showing progressive increases in ash content. In each of these examples, the product in the first zone is contaminated with dirty slimes which have run down the edge of the table. This is especially

	Issaquah Coal. Raw Slack, to $\frac{1}{16}$ in.	
	Weight per cent.	Ash per cent.
Zone No. 1	7·1	14·6
„ 2	18·7	6·2
„ 3	10·2	7·8
„ 4	4·5	8·2
„ 5	13·2	9·9
„ 6	46·3	34·2

	Wilkeson Coal. Jig Washery Refuse.	
	Weight per cent.	Ash per cent.
Zones 1 to 4 —		
Through 20 mesh . .	2·4	43·7
On 20 mesh . . .	2·0	9·6
Zone No. 5	7·4	11·6
„ 6	5·3	12·6

marked with the test on Wilkeson washery refuse where the through 20 mesh material in the bulk sample from the first four zones contains 43·7 per cent. of ash and the over 20 mesh material contains only 9·6 per cent.

Slurry.—The treatment of the slurry produced in jig washers on special "slime" tables is worthy of thorough investigation in this country. In America, where the coke ovens are not situated at the collieries, slurry is frequently thrown away because it is of no value. In England, however, the recovery of the coal particles in slurry for supply to the coke ovens is desirable, but, for this purpose, the slurry must be clean.

As illustrating the possibilities of slurry treatment by tabling the experiences of the Renton Coal Co., of Washington State, U.S.A. (*Coal Age*, 1921, April 28, 741), on the washing of slurry on Deister-Overstrom tables, and of the Allerdale Coal Co., Ltd., Cumberland, on H.H. concentrating tables are interesting. At Allerdale the slurry contained between 35 and 37 per cent. of ash, and this was reduced to considerably under half.

At Renton, five tables were erected to treat a pond containing 200,000 tons of sludge. The raw material contained 28 to 30 per

cent. of ash, and this could be reduced on the tables "to 10 per cent. without losing an appreciable amount of clean coal in the refuse."

Fig. 138 is a photograph of a Deister-Overstrom table treating material from $\frac{1}{16}$ in. to 0. In operation, this particular table at Coaldale, Pennsylvania, treats the underflow from a settling tank. The feed consists of material containing 23 to 26 per cent. of ash, which is separated into washed coal with about 17 per cent. of ash and refuse with about 60 per cent. The capacity on such a feed is 5 tons per hour.

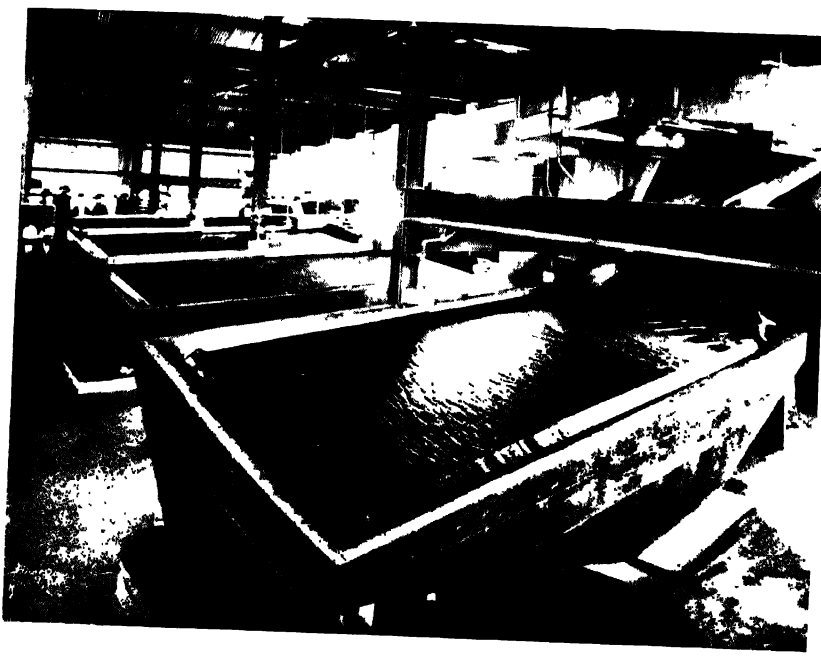

FIG. 138.—Deister-Overstrom Table Treating Settlings from Water Clarifying Tank.

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CHAPTER XVII

DRY CLEANING: PNEUMATIC JIGS

THE EFFECT OF WASHING WATER IN WASHED COAL USED FOR VARIOUS INDUSTRIAL PROCESSES

ONE of the commonest arguments in favour of cleaning coal is that part of the cost of handling and transporting dirty coal is a direct loss to the consumer because the mineral matter in coal has no value. The water remaining in coal after washing is equally valueless, and all the cost of handling and transporting the water in wet coal is a direct loss to the consumer. On this score, indeed, there is no advantage whatever in reducing the ash content of a coal by cleaning it and simultaneously increasing its moisture content by an equivalent amount. Assuming the same efficiency of refuse removal, at the same cost, dry cleaning holds a considerable advantage over wet washing on the grounds of the cost of delivery to the consumer. Unfortunately, on the smaller sizes of coal, dry-cleaning processes are not so efficient as wet processes.

Apart from the factor of transport cost, the use of dry coal for many industrial purposes is preferable to the use of wet coal. The greatest advantage may be in the coking industry. In Great Britain and Continental Europe almost all the coal used for coking purposes is previously cleaned by a wet-washing process. Washed coking slack (less than, say, $\frac{5}{8}$ in. in size) on drainage for twenty-four hours in bunkers will still contain about 10 per cent. of moisture. The slurry, which is usually mixed with the coking slack and charged to the ovens, will retain about 30 per cent. of water.

It may therefore be assumed that, on the average, washed slack contains at least 10 per cent. of added water, and that, in a normal 21 in. oven of 10 tons capacity, at least 1 ton of water is charged with the coal. This water is evaporated and the resulting steam superheated to, say, 200° C. before leaving the oven. For this purpose 2,688,000 B.Th.U. are required. It is known that, in a wide (21 in.) oven, the rate of supply of gas for heating is approximately 1,250 cubic feet per hour, the gas having a heating value of about 500 B.Th.U. per cubic foot. With a coking time of thirty-two hours, 20,000,000 B.Th.U. is generated in the flues. Of this quantity it is known that only about 50 per cent. passes through the oven walls and into the charge during the upstream and down-stream flow of the burning and burnt gases. The removal of the

washing water therefore requires $\frac{2,688,000 \times 100}{\frac{1}{2} \times 20,000,000}$ or nearly 27 per cent. of all the heat supplied to the coal charge. In a coking time of thirty-two hours this is equivalent to eight and a half hours for water removal alone. A reduction of the water from 10 to 3 per cent. would reduce the coking time by six hours.

In more modern ovens, using higher flue temperatures and greater rates of gas supply (5,000 cubic feet per hour), the coking time is about sixteen hours, and charges of 14 tons may be used, containing 1·4 tons of water. Of the 20 million B.Th.U. passing into the charge, about 19 per cent. is used up in removing the moisture, and this is equivalent to about three hours of the coking time.

The effect of the added washing water is not only to increase the coking time, but also to reduce the amount of surplus gas available and to introduce additional sources of expense. In the example given, 10,752 cubic feet of fuel gas are wasted in removing washing water from the wider oven; if this were not necessary the amount of surplus gas available would be increased from, say, 50 per cent. to 62 per cent. of the total (assuming 10,000 cubic feet of gas per ton of coal), with a possible corresponding increase in revenue. The washing water leaving the oven as steam must be cooled in the condensing plant, and in the direct and semi-direct processes of ammonia recovery, this demands an increased cooling surface and an enhanced consumption of cooling water. The washing water, after condensation, appears in the ammoniacal liquor, and must be elevated and stored before passing to the liquor stills. Here it is reheated nearly to boiling-point. Finally, this large bulk of water, freed from ammonia, creates a further problem in its disposal on account of its phenol content. Such a "noxious" effluent cannot be run into rivers, and special means must sometimes be employed for its purification.

In the direct ammonia-recovery process the water is not condensed until after the ammonia has been extracted, but the steam must eventually be condensed and the condensed water is still classed as a noxious effluent because of its naphthalene content.

There are, however, certain advantages in the use of wet coal in coke-oven practice, for the ammonia yield is said to be increased by several pounds per ton of coal. As, however, the market value of sulphate of ammonia is declining owing to the success of the synthetic ammonia process, this factor is becoming of decreasing importance. Moreover, the benefit of a higher yield of ammonia is more than counterbalanced by the fact that the lower temperatures in the oven, caused by the presence of water, result in reduced rates of heating and a decrease in the yields of gas, tar and benzol.

The use of dry coal, however, introduces several practical difficulties in coke-oven operation which do not arise, or do not so readily arise, if the coal charged to the ovens is wet. With dry coal there is a much greater formation of retort carbon, and fine

dry coal tends to be carried up the ascension pipes, causing trouble with stopped pipes, and with the valves. In extreme cases, fine dry coal collects in the hydraulic main, increases the difficulty of keeping it clear, and contaminates the tar. Salty coals are not so harmful to the walls of a coke oven if they are dry as they are when charged wet, but this advantage is partly outweighed by the fact that, during wet washing, some of the salt is removed from the coal by solution. Moreover, the quality of the coke may not be so satisfactory when dry coal is used. For example, when dry coal of high volatile matter content is coked in very narrow (14 in.) ovens, the coke tends to be spongy at the centre of the oven, whereas wet coal produces a satisfactory coke.

The disadvantages of using wet coal, although serious in coke-oven practice, are still more pronounced in the manufacture of coal gas, so that only a small proportion of the coal used in horizontal retorts is washed. If gas coal for carbonisation in horizontal retorts could be efficiently cleaned without wetting, the range of coals available for use would be extended, the resulting coke would be more saleable, and the output of gas per retort would be increased. The reduced amount of pan ash for disposal would also effect a saving, and in some gasworks situated in cities and large towns the saving would be considerable.

The presence of large amounts of water in coal is also a disadvantage for other industrial processes, for example, in boiler firing, either by hand or mechanical stoking or by pulverised fuel. The evaporation of the water reduces the fuel efficiency, and in the use of pulverised fuel the moisture may have a harmful effect on the brickwork of the combustion chamber, and usually the coal must be dried before it can be satisfactorily pulverised. In metallurgical furnaces a high content of steam in the atmosphere of the furnace frequently has a harmful effect, being in part responsible for the scaling of billets in a reheating furnace.

In the actual process of the removal of refuse from coal there are several advantages of air as the separating medium rather than water. In the first place, the availability of the supply of the separating medium offers no difficulties, and, if the air can be satisfactorily filtered to remove the dust particles, no question of its disposal arises. In many British and other collieries the supply of water is not abundant, and the disposal of the waste washery water is not always easy. Air-cleaned coal has a bright, clean appearance; water-washed coal does not always preserve its lustrous appearance unless it is sprayed with clean water. The greatest difficulty experienced in wet washing, however, is the formation of slurry. Slurry is always a troublesome material, and when the coal is of inferior coking quality its disposal raises a difficult problem. If the coal washed is a good coking coal, the slurry is usually mixed with the washed coal and charged to the ovens, this being thought to be the easiest method of disposing of it. Unless it is washed in a

special slurry washer, slurry usually contains about 20 per cent. of ash and 30 per cent. of water, and much of the remaining material consists of non-coking fusain. Slurry cannot be evenly mixed with the coal and the aggregates of slurry in the oven charge increase the coking time, do not contract from the oven walls, thereby causing " stickers " and interfering with pushing schedules, and the coke formed is weak and crumbles to breeze.

On the other hand, the dry dust separated from coal before or during dry-cleaning is not easy to collect. Low pressure fabric filters have been used with some success, but it would seem desirable to use some form of dust separator employing a closed system with re-circulation of the air. The disposal of the dust, which is not cleaned, is also a matter of difficulty unless the colliery is equipped with pulverised fuel boilers. It must otherwise be transported in sacks or in tank wagons and sold in a market in which, at present, there is a limited demand. It would seem, indeed, that the collection and disposal of dry and dirty coal dust is, at present, as difficult a matter as the disposal of dirty slurry, and a greater difficulty than the disposal of clean slurry, which, well mixed with a coking slack, does not materially impair the quality of the coke.

It is difficult to place a monetary value on the advantage of dry rather than wet cleaning, for it depends entirely on the nature of the coal and its ultimate use. With nut coal, from which the water can drain almost completely, there is little, if any, advantage, and the choice between a wet and a dry cleaning process depends upon the cost and efficiency of the operation. With small coal, however, there may be, for the majority of purposes, a certain advantage, but in the present state of our knowledge, a true comparison of the relative values of dry and wet coals for coke manufacture cannot be given. For boiler or furnace heating, if washed smalls and slacks are used an excess moisture content reduces the heating value of the fuel by more than the percentage of water present, for the water is not only itself waste, but it requires heat to evaporate it and bring it to the flue gas temperature.

In certain parts of America, where the winters are cold and the coal must travel some distance by rail, a truck (or car) of coal which has been washed by a wet process arrives at its destination as a frozen mass, and to unload it is a very difficult problem. It is so difficult that some mines can only ship their coal in the summer, and others confine their sales during the winter to markets that will take unwashed material. In certain other States where it is necessary to clean the coal, there is no water available for the purpose, and dry cleaning is again a necessity.

DRY CLEANING PROCESSES

Appliances for the dry cleaning of coal were first introduced about 1850, and since that date a variety of methods has been employed. These may be classified into four groups, viz. :—

1. Methods employing stationary devices with continuous or intermittent air currents.
2. Methods employing reciprocating devices with continuous or intermittent air currents.
3. Methods in which separation is effected by virtue of differences in coefficients of friction.
4. Methods in which separation is effected by centrifugal force.

There are certain dry processes of separation of dirt from coal which do not fall strictly into any one of these four classes. Certain of them (*e.g.*, spiral separators) depend partly on centrifugal force and partly on friction, others (*e.g.*, the process recently invented by Messrs. W. H. and S. R. Berrisford) depend partly on friction and partly, it is stated, on the greater resiliency of coal than of shale These processes will be described under the third group, for it is probable that the influence of friction plays the greatest part.

STATIONARY DEVICES USING AIR CURRENTS

Included in this class are two types of appliance, namely, pneumatic or air jigs, employing an intermittent air current, and devices employing a continuous air current in either a vertical or a horizontal direction. In an air jig the pulsating air current causes a stratification of the bed through which the air passes. In horizontal or vertical blowers, a continuous current of air crosses a stream of particles projected in a direction normal to it and causes a differential displacement of the light and heavy particles. Neither pneumatic jigs nor blowers have been used extensively either in coal cleaning or ore dressing, but each type has been used to a certain extent.

The theory of pneumatic separation is identical with that of separation in water except that the different specific gravity of the medium must be taken into account. The theory of separation of coal from dirt in a medium of water was described at length in Chapters III and IV. It was shown that an upward current of water would maintain a given particle in suspension if the speed of the water current were equal to the terminal velocity of fall of the particle in still water. Similarly, a particle may be supported by an upward current of air if the air has the same speed as that at which the particle would ultimately fall in air (a speed which is not infinite on account of the viscous resistance of the air). In an air jig, an air current is forced intermittently through a bed of particles and the coal particles are unable to fall against the current. The dirt particles, being heavier, are able to work their way to the lowest layers. When stratification has thus been set up, various means may be employed to separate the layers.

In Chapter IV it was stated that the phenomenon of hindered settling probably has a considerable influence in determining the speed of water current necessary to cause separation and the ratio

of sizes that can be separated. Its influence is probably equally important when air is used as the separating medium, but at present the phenomenon is not understood and it is impossible to evaluate its effect numerically. It is, however, known that the feed must be more closely sized when a pneumatic jig (or dry cleaning table) is used than when separation is effected by similar appliances using water as the separating medium, and the ratio of the degrees of sizing necessary is suggested by the formula

$$\frac{r_1}{r_2} = \frac{s_2 - s}{s_1 - s}.$$

In water, $s = 1$ and $\frac{r_1}{r_2} = \frac{s_2 - 1}{s_1 - 1} = 5{\cdot}0$ for dirt (S.G. 2·5) and coal (S.G. 1·3). The specific gravity of air is relatively negligible, and putting $s = 0$, $\frac{r_1}{r_2} = 1{\cdot}92$.

The Krom Pneumatic Jig.—The Krom jig was first introduced in the United States and was patented in England in 1874 It was

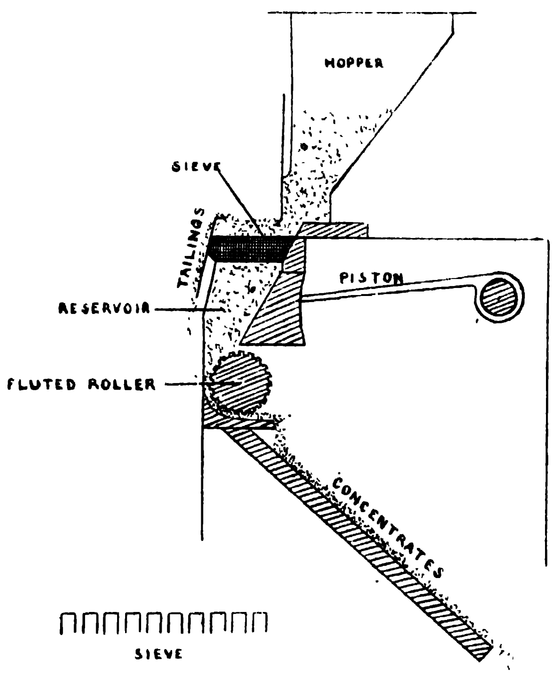

FIG. 139.—The Krom Pneumatic Jig.

used for the treatment of ores which tended to assume the form of thin plates, and which, especially if not properly wetted, floated across the water on the top of the bed. It has also been used in New Mexico for coal cleaning.

It consisted of a sieve, upon which the feed was separated into layers of light and heavy particles by the action of a pulsating air current passing upwards through it. The jig is shown diagrammatically in Fig. 139. The ore was fed from the hopper on to a sieve composed of tubes of wire gauze. The wire gauze tubes were open at the bottom but perforated on the top and sides, and were spaced apart. The feed worked its way into the interstices between the tubes, where the intermittent air current caused its lighter constituents to pass upwards. The concentrated ore passed between the tubes into the reservoir, which was maintained full of material. The material in the sieve therefore rested on the bed of concentrate in the reservoir, and only the heaviest of its particles could fall down into the reservoir. The discharge from the reservoir was controlled by the rotation of the fluted roller.

The air current was produced by the oscillation of a piston about an axis at a rate of about 400 strokes per minute. The capacity of the jig was about ½ ton per hour. Although the Krom jig is said to have been efficient, it has not found great favour because of its low capacity and because it required a large volume of air.

The Plumb Pneumatic Jig.—This appliance is illustrated diagrammatically in Fig. 140. The jig consists of a cast-iron trough, 2 to 3 ft. long and 3 in. wide, fitted with a sieve, upon which the material rests, and an air chamber beneath the sieve. Air impulses are created by a fan, the current being admitted intermittently by means of a rotating valve. The pulsations number 400 to 500 per minute at a pressure of 10 to 30 lb. per sq. in. at the valve, the higher pressures being used for larger material.

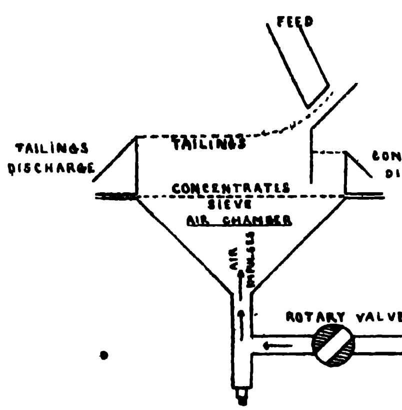

FIG. 140.—The Plumb Pneumatic Jig.

The feed, which requires rather close sizing, is supplied by a shoot at one end of the box and the air impulses cause it to stratify with the lighter material in a layer uppermost. The lighter particles are discharged over a lip along one side of the box; the heavier material passes under a vertical gate running longitudinally along the box, with its lower end slightly above the sieve, and is discharged over a lip along the other side. The lip on the tailings-discharge side is higher than the lip on the concentrates-discharge side, so that the levels on each side of the gate are balanced.

The air chamber has sloping sides, forming a tapered vessel, into

which some of the finest ore particles fall. They are discharged from bottom openings by removing stoppers at intervals. The air is admitted from two mains along one side of the box.

The Plumb jig has been used in America for the dressing of ores passing a 12-mesh screen (about $\frac{1}{20}$ in.); with material of this size, a box 3 ft. long and 3 in. wide had a capacity of under 1 ton per hour. Close sizing is required for efficient operation, but the jig is said to be incapable of treating material above about $\frac{1}{10}$ in., and to be unsuccessful with very fine materials.

The Paddock Pneumatic Jig.—The Paddock pneumatic jig, shown in Fig. 141, was introduced in the United States about 1888. It was similar in principle to the Krom and Plumb jigs, stratification being produced by means of an intermittent air current forced through a perforated plate, upon which the material rested. The current was produced by a bellows worked by two eccentrics running at 450 r.p.m.

The Paddock jig had several features resembling a table rather than a jig. Indeed, a modification of the Paddock appliance, which is similar in principle and differs only in details (the Hooper-Paddock air-jig) is often called a pneumatic table.

The sieve or perforated deck on to which the material was fed consisted of an iron grating or grid, over which a piece of stout broadcloth was stretched. On the broadcloth, a diagonal brass grating rested, giving the deck a riffled appearance. The strips of this grating were about $\frac{3}{8}$ in. high. Above them a second brass grating was placed, the strips of the upper grating being almost at right angles to those of the lower grating and about 2 in. high.*

In the Hooper-Paddock jig, the lower set of brass strips are $\frac{1}{8}$ to $\frac{1}{4}$ in. high, $\frac{1}{12}$ in. wide, and spaced $\frac{5}{8}$ to $1\frac{1}{4}$ in. apart. The upper set of strips are $3\frac{1}{2}$ in. high, $\frac{1}{12}$ in. thick and $\frac{5}{8}$ to $\frac{3}{4}$ in. apart. The two sets are set diagonally to each other and at angles of 30 to 45 degrees with the side of the frame.

The feed of coal or ore is supplied to the surface from a hopper at one end of the deck, and the products are discharged over a lip at the other end. The blasts of air through the deck cause the feed to stratify; the heavier material, as it travels over the deck, passes between the strips of the lower grating to one side, the lighter material is carried by the upper riffles in a direction almost at right angles to that of the heavy material, to the other side. Adjustable partitions along the discharge sides enable the products to be divided as required. The Bonson table used in America for ore-dressing has a similar arrangement of riffling.

The original Paddock appliance was said to work fairly well on particles between about 35 and 120 mesh fed at the rate of half a ton an hour, but it did not work successfully with material outside these sizes. The Hooper-Paddock jig, however, spreads the feed out over

* *Eng. and Min. Journal*, 1892, 54, 130.

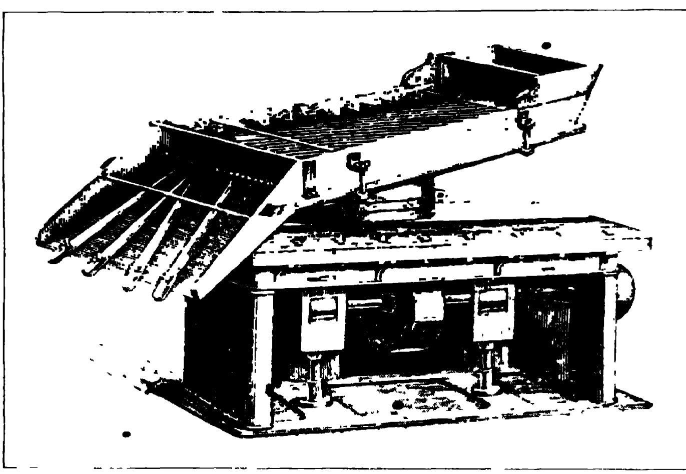

FIG. 141. The Paddock Pneumatic Jig

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a considerably wider lateral area, and particles of size from 2 mm. to 0 can be treated if subjected to a preliminary sizing. Its capacity is about half a ton per hour.

The Card Air Concentrator.—The Card concentrator, invented in America in 1891, was an air-jig used for ore dressing. The ore was supplied from a hopper to an inclined trough. The floor of the trough was composed of wire gauze and was divided into sections to allow the lower layers of material to be removed. The air-chamber under the floor of the trough was divided into a series of boxes, which were placed in a row with a gap between each pair and the wire gauze formed the top of the boxes. Between each pair of boxes there was therefore a gap down which the concentrated ore could fall.

The air was supplied intermittently through the sides of the boxes from a bellows actuated by a beater. The air passed through the perforated tops and stratified the bed. The heavy particles falling through the gaps were discharged by a special rocking mechanism at the base. The rocker was driven by a ratchet and pawl, and the rate of discharge was controlled by the rate of operation of the rocker.

The Kirkup Process.—The Kirkup table, recently introduced in England, is essentially a pneumatic jig, but incorporates several novel features. The first experiments were carried out in 1924 by Mr. R. Dickinson at Armstrong College, Newcastle-on-Tyne, and on a larger scale by Messrs. R. H. Kirkup, of Gateshead-on-Tyne.

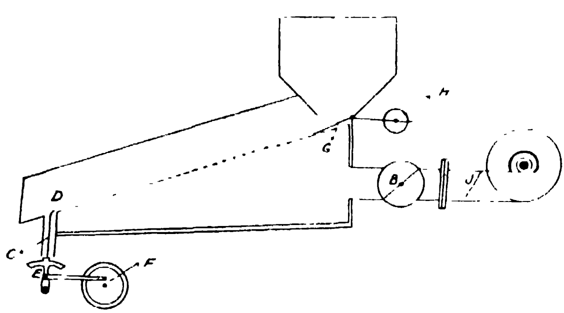

FIG. 142.—The Kirkup Table: Diagram.

The first commercial unit was put into operation at Consett, Co. Durham, in January, 1928. The raw coal is fed from a hopper on to a perforated plate inclined at 12½° to 15° to the horizontal. A pulsating air current supplied to the underside of the inclined plate stratifies the material on the screen and the layers of material so formed are separated at the lower end.

In operation, the raw coal is supplied at such a rate as to allow a bed about 6 to 8 in. deep to be established. The intermittent air current keeps it in a sufficiently loose condition to enable heavy

particles to pass downwards to the bottom of the bed, at the same time moving the lighter particles into the upper layers. The air current is supplied to an air-tight chamber, A, Fig. 142, at a pressure of 3 to 6 in. water gauge. The fan creating the air current is continuous in action, but the entry of air into the chamber is permitted periodically by a rotary or reciprocating valve, B. The number of pulsations is about 200 per minute, the pulsator being driven separately. The bed plate is $\frac{1}{16}$ in. thick, with apertures spaced $\frac{1}{8}$ in. between centres.

When the stratified material reaches the bottom of the inclined plane, the lowest layers are removed through a slot, D, in the base into a vertical shoot. The shoot is divided into two compartments by the plate, C, so that only the heaviest particles pass down the first compartment of the shoot and intermediate material (middlings) passes down the second compartment. At its upper end the dividing plate, C, is bent over as shown in Fig. 142. Below the shoot, an oscillating segment, E, was first arranged, to assist the delivery of material from the shoots, and to enable the rate of removal of refuse and middlings to be controlled. This is shown in Fig. 142. In later plants, the oscillating device has been replaced by five-pronged star extractors, which may be driven at different speeds by a variable gear as in Fig. 143.

The drive is communicated to each extractor by a rubber-faced pulley bearing on a flat disc, contact being ensured by a spring behind the disc. The rubber-faced pulley is drawn nearer to the centre of the disc to increase the speed of rotation of the extractor and nearer to the circumference to reduce the speed. There are separate drives for the middlings and refuse extractors, but the adjustment of each is controlled by the operation of a single hand-wheel through dog-clutches. The adjustment is therefore simple

The plant for industrial operation is shown in Fig. 143. The inclined plate is 14 ft. long and 2 ft. wide, and is designed to treat 25 tons of coal per hour, the coal being unsized coal through a $1\frac{1}{2}$ in. screen. The plate is encased by sides 12 in. high. The air current is supplied by a fan capable of delivering varying quantities of air to suit the coal treated. The pulsations of the air current are created by a rotating valve with two blades, and about 200 pulsations are made per minute. A flap valve is placed between the fan and the pulsator to vary the air supply to the table.

The plant is of simple design, and may be constructed cheaply Duplication, to give larger capacities, offers no difficulties, and the operating cost should be small. Since the air current is only supplied to one table during half the revolutions of the valve pulsator, two tables may be supplied with air from one fan as in the Raw process (p. 361). The two blades of the double pulsator are then inclined at right angles to each other, and when the air current is cut off from one table it is supplied to the second. In this way no more power is required to supply the air current for two tables than for one.

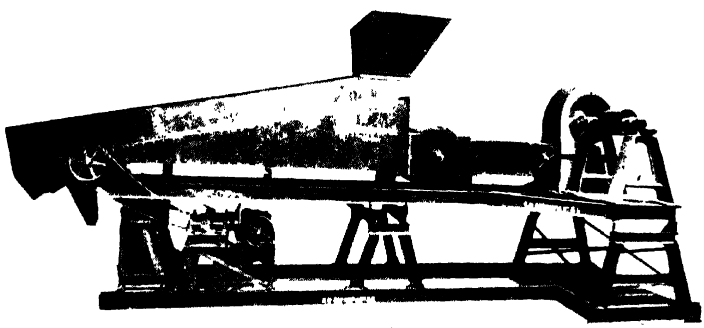

FIG. 143.—The Kukup Table

[To face page 336.

The process has not yet been in operation on a commercial scale for a sufficient length of time to prove its suitability and efficiency, but it offers the advantage over other present processes that it can treat unsized material. It is said, indeed, to work more satisfactorily with an unsized feed than with sized material. Fluctuations in the rate of feed, or in the proportions of coal and dirt, may present some difficulties, for it is improbable that the stratification of the material will be perfect and an unusually high proportion of dirt may result in inefficient operation. Similarly, an occasional batch of coal containing a small proportion of dirt may result in a loss of coal with the refuse. The safeguard against imperfect separation is the re-circulation of a large proportion of the feed in the form of middlings, which reduces the throughput of the table.

With its deep bed, the action of the table may be more sensitive to the presence of free moisture in the coal than are other modern dry-cleaning tables which work with a thin coal bed. The resistance

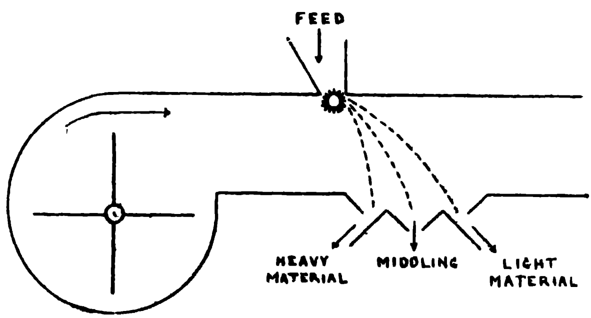

FIG. 144—Diagram of Action of Horizontal Blower.

to the passage of air is already high, and the tendency of moist particles to adhere to one another will prevent the free circulation of air around them. Wet coal cannot be treated on the table for this reason.

When the supply of coal to the table is irregular, the plant is shut down automatically when the raw coal hopper is empty, in order that the bed of dirt resting on the perforated plate shall not be lost. The maintenance of a proper dirt bed is important in the operation of the table, as without it coal may be extracted by the first star extractor.

Horizontal and Vertical Blowers.—Blowers creating horizontal air currents have been used in several forms for ore dressing. The principle is illustrated diagrammatically in Fig. 144. A continuous air current produced by a revolving fan blows across a stream of particles falling vertically. The lightest constituents of the feed are deflected from their normal course to a greater extent than the heavier material, and a separation is therefore possible. Such machines are useful where the constituents of the feed are of widely

different shape or density, and have been employed for the recovery of alluvial gold and diamonds. They are also applicable to the recovery of thin flakes of graphite from graphitic sands and of wheat from chaff.

The Hochstraate Machine.—An appliance using this principle was employed for coal cleaning in England about 1860, and another, the Hochstraate machine, was used at the Zollverein and Rheinpreussen Collieries, in Germany, about 1880 (*Zeit. für Berg., Hütte und Salinenwesen*, 1882, 30, 280 ; 1887, 35, 265 ; 1894, 42, 235).

The raw coal was sized before treatment, and as it slid over the end of a shoot it encountered a blast of air. The air caused the coal and smaller shale particles to travel up inclines where a baffling system stopped the flat shale and allowed the coal to go forward. The shale and the largest coal particles, which could not be driven up the inclines, fell together into a trough and were separated by a wet method.

The Mumford and Moodie Apparatus.—The device shown diagrammatically in Fig. 145, and modifications of it, have been employed extensively for the removal of dust from coal and ore and of sand from asbestos. Although primarily a de-dusting device, with accurately sized material it functions as a concentrator for fines.

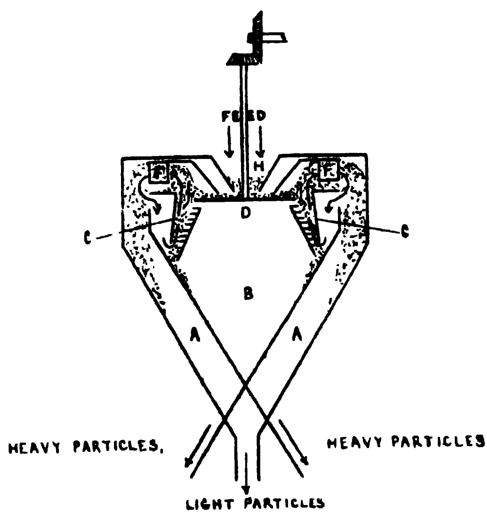

FIG. 145.—The Mumford and Moodie Separator.

The feed is supplied from a hopper, H, on to a rapidly revolving disc, D, which throws it off laterally in the form of a spray to the surrounding ring, C. An air current is induced by the fan blades, F, in an upward direction through the space between the disc, D, and the ring, C, and carries the smallest particles upwards with it. These small or light particles pass into the outer vessel, A, whereas the large or heavy particles fall into the inner vessel, B. The air current returns in the direction shown by the arrows, to be used over again.

The machine is self-contained, and the air current being continuous no loss or trouble is experienced by the escape of dust. Its action is readily adjustable by varying the speed of the drive, but since the fan and the disc are driven directly by the same shaft, they are not capable of individual adjustment. It is made in varying sizes, one 4 ft. in diameter dealing with about 4 tons of feed per hour.

The importance of this and similar devices lies in their use for dust removal rather than as methods for cleaning. They might serve as useful accessory appliances in many coal-washing plants where the finest dust complicates the washing process.

In another device, similar in general design to the Mumford and Moodie machine, which has also been used for separating the dirt from coal, the feed was distributed laterally by a rapidly revolving disc mounted horizontally and suitably driven. As it flew off

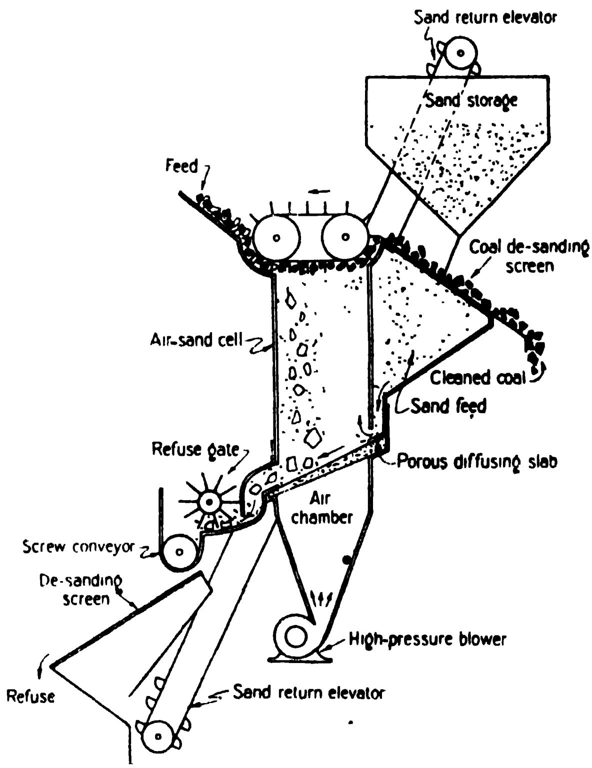

FIG. 146.—Proposed Arrangement of Air-Sand Process (Frazer and Yancey).

the disc as a spray, it passed across an annular opening and an air current played vertically downwards across it. The light particles of coal were affected by the current and passed through the annular space, whereas the heavy dirt particles were relatively unaffected and passed across the opening. The machine was, however, never very successful in practice, because much of the coal passed across the aperture with the dirt, and the dirt, because of its higher coefficient of friction, left the disc at a lower speed than the coal and fell through the gap. It might be worthy of consideration as a separator of coal and dirt for a coal with a higher coefficient of friction than the

shale associated with it, examples of which exception to the usual rule are known.

The Frazer and Yancey Process.—Frazer and Yancey (Amer. Inst. Min. [illegible], Feb. meeting) have devised an ingenious method to effect a density separation of coal and refuse by forcing streams of air through a vessel containing sand, so that a dry fluid medium is obtained with a bulk density of 1·45. Coal floats across the containing vessel; refuse, because of its greater density, sinks through the mixture. The separation is independent of the relative sizes of the coal and refuse particles.

Fig. 146 is a diagrammatic cross-section of the suggested arrangement of a large-scale plant. The plant consists of a main cell in which the air-sand mixture is contained. The air is forced from an air chamber through a porous diaphragm and passes through the sand in a multitude of fine streams. The coal is fed by a rotating bladed wheel at one side of the cell; the clean coal is collected at the opposite side, and the refuse is discharged through a gate at the bottom of the vessel. Both products are de-sanded and the sand is kept in circulation by an elevator and storage hopper.

The device is not yet in operation on a commercial scale, and the design is based on the experience of small scale experimental work, which has yielded promising results. The small-scale plant consisted of a cast-iron cell, divided by a porous plate into air and sand chambers. Considerable difficulty was experienced in obtaining the most suitable material for use as the diaphragm, but porous concrete slabs or "Filtros" plates were found to give satisfactory results. In a cell 10 in. square, a steel plate with $\frac{1}{64}$ in. perforations spaced $1\frac{1}{2}$ in. apart, with extra holes along the sides and in the corners, was found to work satisfactorily. River sand was employed, and air was supplied at a pressure of $1\frac{1}{2}$ to 3 in. of mercury (20 to 40 in. of water) in the air chamber. The pressure used was just sufficient to cause streams of air to flow through the sand and give the fluid mixture the appearance of continuous "boiling." The mixture was then easily penetrated or stirred by the hand, and a spindle hydrometer placed in it gave a reading of S.G. 1·45.

The efficiency of the separation effected in the laboratory model can be gauged by the results of the following test on nut coal (2 to 3 in.):—

	Yield per cent.	Ash per cent.
Raw coal floating in liquid S.G. 1·45	90·4	9·9
Clean coal by air-sand process	87·3	9·9
Coal washed in jig	82·6	10·3

These results show an improvement on the jig practice, but, for a product with 9·9 per cent. of ash, the yield is 3 per cent. less than the theoretical maximum.

For the plant shown in Fig. 146, the cell is 1½ to 2 ft. deep, and contains 20-mesh river sand. The porous diaphragm is inclined towards the refuse-discharge side, being thinner on that side, to allow for the increased resistance offered by the deeper bed.

The process is simple, and, if the practical difficulties can be overcome in a large-scale plant, is assured of success because of its ability to deal with an unsized feed. Its scope is, however, limited to coal above a certain size. If the sand used passes through a 20-mesh sieve and is recovered by passing the coal over a de-sanding screen, it follows that coal below 20 mesh cannot be recovered and must be removed before cleaning. The shale must also be below a certain size or it would be supported dynamically with the sand.

CHAPTER XVIII

DRY CLEANING: PNEUMATIC TABLES

PNEUMATIC tables are those devices classified in Chapter XVII as reciprocating devices using air currents. It is only within the last decade that they have been used industrially for coal cleaning.

The Sutton-Steele Pneumatic Table.—The first pneumatic table (as distinct from a pneumatic jig) was made by Messrs. Sutton, Steele and Steele. It was used for the cleaning of agricultural produce, seeds, cereals, nuts and beans, the husks or shells and bad or dried matter being removed from the desired product by reason of their lower specific gravity, and for ore dressing in those parts of the United States of America where water is often scarce. It was patented in England in 1905.

Many appliances had been used previous to the introduction of the Sutton, Steele and Steele table, in which stratification of the feed was induced by shaking the surface or by an air current passing upwards through a perforated surface. Devices had also been used in which the material was spread out on a riffled deck. The Sutton, Steel and Steele invention consisted in combining an upward current of air with a shaking movement of the riffled surface.

A modified table, the "C. J.," was used for coal cleaning, and the first commercial installation consisted of twelve "C. J." tables, erected at the Brilliant mine of the St. Louis, Rocky Mountain and Pacific Company in New Mexico in 1919. The sales rights of the Sutton, Steele and Steele table were subsequently acquired by the American Coal Cleaning Corporation, who have modified it as a result of experiments, and have adapted it for the dry cleaning of all kinds of coal. The table is manufactured in this country by the Birtley Iron Company. At the beginning of 1928 the Birtley Iron Company had erected twenty separators, and were installing thirteen others to attain a total capacity of 865 tons per hour.

The early Sutton-Steele table for purifying agricultural produce was somewhat similar in appearance to the Wilfley table. The deck consisted of an approximately rectangular framework of longitudinal wooden strips upon which rested a covering cloth, itself covered with tapering riffles. A blast of air, supplied from below the deck, was distributed fairly uniformly by the wooden strips and stratified the material on the deck. The lighter material passed transversely across the table over the riffles, and the heavier material travelled longitudinally between the riffles by reason of the motion given to the deck by an actuating mechanism. The mechanism consisted of

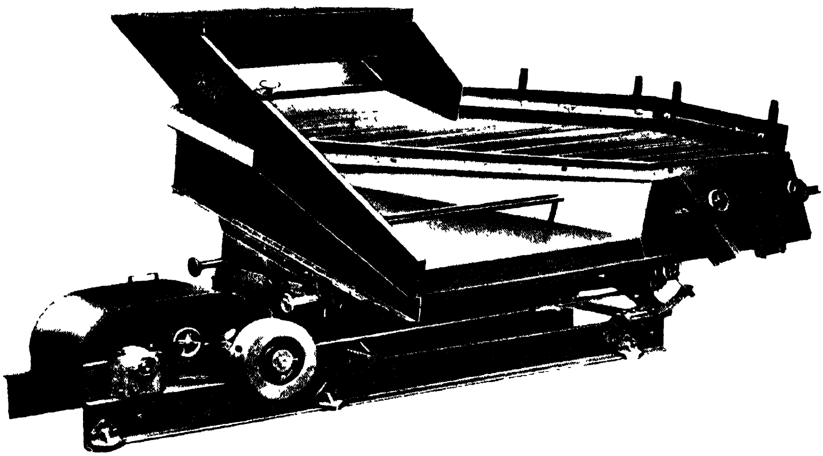

FIG. 147.—The S. J. Separator: Back View.

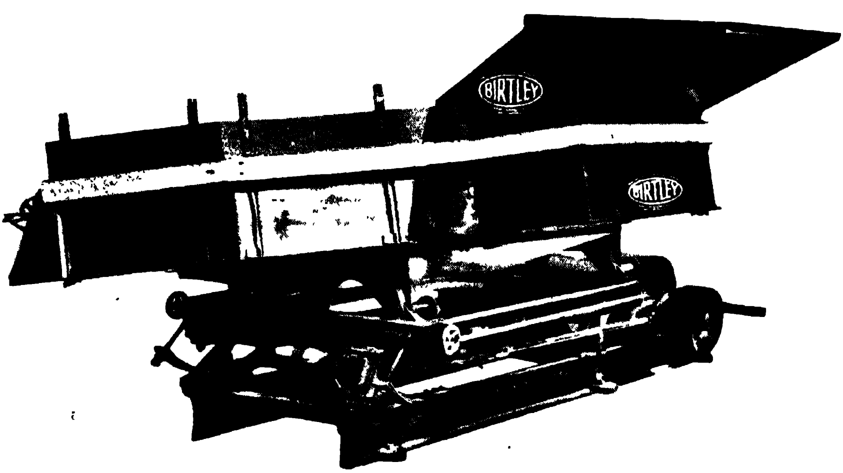

FIG. 148. The S. J. Separator: Front View.

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a cam working against a bell-crank lever, and compressing a helical spring during the forward stroke. When the table was at the end of its forward stroke the lever was released and the spring jerked the table sharply backwards.

The arrangement of the deck differed somewhat for different purposes, some models having an unriffled deck and being supported on flexible spring legs arranged as in the H.H. wet concentrating table. The tables which are now made by the American Coal Cleaning Corporation and the Birtley Iron Company for coal cleaning have, however, been modified from the earlier C.J. table both in shape and size, and an improved actuating mechanism is employed.

The S.J. Pneumatic Separator.—The S.J. table was the first recognised improvement on the C.J. It is illustrated in Figs. 147 and 148.

The deck of the old Sutton-Steele table was rectangular in plan, but it was noticed that part of the surface was of little use for coal-cleaning, and the shape was therefore altered to that of the present S.J. table. In effect, two corners were cut off, the bottom left-hand corner and the top right-hand corner. The removal of the top right-hand corner, and the fixing of a barrier across it in the path that would otherwise be taken by some of the refuse, serves a special purpose, to which reference will be made later.

The surface upon which the separation is effected consists of square-mesh wire gauze, the size of the holes varying for the different sizes of coal. It is overlaid with wide square-mesh chicken wire, in order to give a grip on the surface and prevent the particles from sliding about on it. The holes in the chicken wire are about 1 in. square, but those in the wire-gauze surface are much smaller and vary according to the size of the coal treated. The usual sizes are as follows:—

Size of Coal.	Size of Holes in Deck.
>1 in.	$\frac{1}{8}$ in. square.
1 in. to $\frac{1}{4}$ in. . .	$\frac{1}{16}$,, ,,
$<\frac{1}{4}$ in.	$\frac{1}{32}$,, ,,

The wire-gauze surface is strengthened with wooden strips (pine) laid longitudinally beneath it, *i.e.*, parallel to the side along which the products are collected. The wooden strips are 2 to $2\frac{1}{2}$ in. high and spaced about 1 in. apart. They serve as support for the deck and also as a bed to which the riffles may be fastened. The riffles themselves are made of light galvanised sheeting, bent at right angles at the bottom to receive the holding screws (or tacks). They taper

from the head to the refuse end, and are laid parallel to the discharge side of the table, terminating along a line roughly parallel to the right-hand or refuse end of the table. The heights of the riffles vary from 2 to $1\frac{1}{4}$ in. at the mechanism end of the table, according to the size of coal treated, and taper to a height of $\frac{3}{32}$ in. at the refuse end. They are spaced 1 in. apart for the smallest sizes of coal and further apart for larger sizes. For certain coals they are 6 in. apart.

Underneath the wooden strips laid as supports for the deck and the riffles is an air chamber separated from the actual deck by an air-distributing vane or guide. The air is supplied from a fan to the air chamber, where the pressure is a few (2 or 3) in. water gauge.

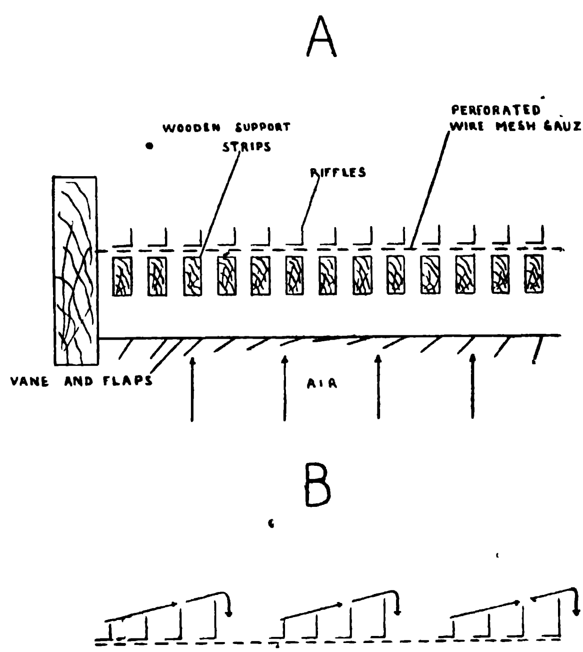

FIG. 149.—Diagram of Deck Construction. S.J. Separator.

The vane through which it passes to the deck consists of a light metal sheet, which is cut in the form of rectangles about 1 in. by 2 in. over its whole area. The rectangles can be bent over by any desired amount, and therefore serve as flaps. Each flap can be opened slightly, to allow only a small quantity of air to pass through the aperture, or more widely, to allow free passage of the air. By opening the flaps by different amounts over the area of the sheet, the air current can be distributed and directed so that it is stronger at some parts of the deck than at others. Fig. 149 (A) is a diagrammatic section showing the arrangement of the riffles, the wire-mesh surface, the supporting strips and the air-distributing vane.

The riffles on the standard deck are of uniform height at the head end of the table, tapering to a constant height at the refuse end.

For special purposes this arrangement can be modified. For example, certain coals contain quantities of thin flat pieces of shale, which are liable to present their largest surface to the air current and remain on the top of the coal. In these circumstances it is necessary to cause the coal to cascade over a riffle, so that the thin shale particles may turn over and slip through the coal to the deck surface. This is accomplished by stepping the riffles, as shown diagrammatically in Fig. 149 (B).

The table is mounted rigidly on two steel channels, to which the actuating mechanism is bolted. A flexible connection between the air chamber and the air-supply main is obtained by an air-tight fabric jointing. The Mark II. actuating mechanism and frame can be seen in Figs. 147 and 148.

The deck is mounted on a steel framework supported by two inclined plates, each of which rests in two V-grooves on cross-pieces on the steel channels. One of the inclined plates is clearly visible on the left of Fig. 148. The drive consists of two eccentrics and connecting rods, which move the deck backwards and forwards. On the forward stroke the deck rises slightly because of the inclination of the plates, and falls again on the backward stroke. At the reversal from the forward to the backward movement, therefore, the deck surface is, as it were, suddenly withdrawn from under the bed of refuse resting on it, and the refuse continues to move, relative to the deck surface, towards its discharge point.

The curved leaf springs shown in Figs. 147 and 148 are not an integral part of the mechanism, but serve to take up slackness between the bearing surfaces. They tend to help in hastening the change from the forward to the backward movement.

The eccentric drive is communicated by a variable-speed cone pulley; the usual rate of revolution is 320 r.p.m., and the stroke is $\frac{3}{8}$ in. in length. A more rapid reciprocation is required with small coal than with large coal, and the stroke is decreased slightly in length if its speed is increased.

The inclination of the deck is variable in two directions, from the head end to the refuse end, and from the feed side to the discharge side. Each inclination is obtained by sliding pins locked in any desired position in slotted keepers. For the side to side inclination the deck is hinged about the centre of each end (Fig. 147) and two curved slots are provided along each side. The locking wheels and one of the slots for the end-to-end inclination can be seen at the sides of the front inclined plate in Fig. 148. The flexible fabric joint and air chamber is situated in the centre of the framework.

In operation, the raw coal is fed from a hopper or a shoot by a shovel feed into a distributing hopper along the top of the table. The shovel feed is employed to ensure a regular rate of supply of the raw coal. The air current, assisted by the vibratory movement of the deck, results in a stratification of the feed so that the heavy

particles form a layer on the surface and the lighter particles rest above them. The air current has a slight lifting effect on the light particles which the heavy ones do not experience, and, because of the inclination of the deck downwards towards the discharge sides, the light particles tend to float over the top of the riffles and travel straight down the table. The heavy particles, however, are trapped between the riffles and are jerked forward between them and across the table by the motion of the deck. The overlaid chicken wire increases the friction between the heavy particles and the surface and prevents them from moving to and fro with the surface. They are thus able to move down the slope towards the right-hand side of the table when the deck moves forward to the right, but are unable to move back up the slope when the deck is withdrawn and moves to the left.

The particles of intermediate density either collect with the shale between the first few riffles, or, if they are carried downwards for a short distance with the clean coal, ultimately find their way to the deck surface, the clean coal remaining above them. They are thus trapped between two riffles and, being then unable to pass over the next riffle, are impelled across the table towards the refuse end by the motion of the deck. Here they tend to mount on to the top of the shale, and, as the riffles towards this end of the table are decreasing in height, the middlings particles are able to pass over them and travel downwards towards the discharge side, whilst the true refuse is still moved further across the table. The current of air passing through the coal lifts the majority of the very fine particles, and these are collected in a hood and removed from the air by a system of filtering.

The process of separation is similar to that effected on a Wilfley or other concentrating table. The chief difference is that the coal moves towards its discharge point by gravity (floating on air) instead of by the force exerted by a water current. The motion of particles being easier in air than in water, they travel across the table much more rapidly, and the capacity of an S.J. air table is about 50 to 100 per cent. greater than that of a concentrating table, although the space that it occupies is considerably less. The concentrating table has, however, one advantage, namely, that, because the particles move more slowly, the products are spread out into a thinner layer over a wider lateral distance, and the division into coal, middlings and refuse can be more easily and accurately adjusted. This will be evident from the results given for the Wilfley table (Chapter XV.); with a very gradual increase in the ash content foot by foot round the table, as these results show, the exact maximum ash content of the clean coal can be accurately fixed.

On the S.J. table, the products are all collected along the discharge side, the division into clean coal, middlings and refuse being made by adjustable dividers (Fig. 147). The middlings product consists partly of intergrown particles, but the exact position of the

dividing line between the clean coal and the middlings is usually indefinite, and a quantity of coal must be collected with the middlings to ensure that no high ash particles pass into the clean product. The reason for this is that the product is not widely spread out in a thin layer, as on a concentrating table.

The dividing line between the middlings and refuse can be rather more exactly fixed, because of the banking barrier placed across the top right-hand corner of the table and down the refuse end. The refuse is jerked by the table motion to the ends of the riffles and against this barrier. When it reaches the barrier, the table motion still tends to force it to the right and to thrust it against the barrier. Consequently it banks, or piles up, and middlings or coal particles are unable to become entangled with it. By means of this barrier it is usually found to be fairly easy to obtain a refuse product reasonably free from coal. As has already been explained, it is less easy, however, to obtain a coal product free from high ash particles.

The barrier across the bottom left-hand corner of the table also serves a useful purpose. By piling up a bank of clean coal along it, a thrust is given to the refuse travelling across the table which assists its passage to the opposite end. It also enables the bulk of the small coal, arising through any fracture of the coal during any stage of the process later than the screening, to be collected separately. The smallest coal tends to mount above the larger pieces, and it can therefore pass over the barrier without the larger pieces.

The beneficial effects resulting from the use of these two barriers are considered to outweigh the fact that they prevent the spreading out of the cleaned product. They increase the efficiency of the final separation of coal from dirt, any loss of coal in the middlings being avoided because they are re-circulated. The re-circulation simply involves a slight reduction of the maximum capacity by the amount of middlings re-treated.

The operating capacity of the S.J. table depends upon the amount of middlings and refuse in the clean coal. With a normal British coal containing 15 to 20 per cent. of ash, the capacities on different sizes of coal are given in Table 94. The power requirements to drive the fan, the table and the shovel feed are also given.

It will be observed that the figures in Table 94 give the sizes of coal between limits in the ratio 2 : 1. A sizing ratio of 2 : 1 is theoretically the maximum range of sizes which can be separated in one operation. In practice, however, although a ratio of 2 : 1 is desirable, it is found that a slightly greater ratio can occasionally be employed with a satisfactory efficiency of separation.

The capacities given assume that the coal is fed uniformly. The separator will work efficiently at half its normal capacity, but its operation is unsatisfactory at less than half the normal or more than the maximum.

TABLE 94.—S.J. TABLE. CAPACITY AND POWER CONSUMPTION

Size of Coal. In.	Capacity. Tons per hour.	Power Required. H.p.
4–2	50–60	35
2–1	30–35	25
1–$\frac{1}{2}$	25–30	20
$\frac{1}{2}$–$\frac{1}{4}$	20–25	15
$\frac{1}{4}$–$\frac{1}{8}$	15–20	10
$\frac{1}{8}$–$\frac{1}{16}$	12–15	6

The following results of operation (Table 95) were obtained with a Durham gas coal, the coal below $\frac{1}{16}$ in. being untreated.

TABLE 95.—RESULTS OF CLEANING WITH S.J. SEPARATOR

Size (in.).	Raw Coal.			Clean Coal.		Refuse.	
	Per cent. of Sample Treated.	Ash per cent.	Per cent Sinking at S.G. 1·5	Ash per cent.	Per cent. Sinking at S.G. 1·5	Ash per cent.	Floating at S.G. 1·5
2–1$\frac{1}{2}$ . .	13·48	12·5	12·6	4·73	1·1	65·25	1·5
1$\frac{1}{2}$–1 . .	16·55	7·32	7·2	4·26	0·9	67·71	1·2
1–$\frac{1}{2}$ . .	26·00	7·9	7·7	3·45	1·4	66·05	2·5
$\frac{1}{2}$–$\frac{1}{4}$ . .	16·66	8·27	8·0	2·82	0·8	67·95	0·3
$\frac{1}{4}$–$\frac{1}{8}$ . .	15·61	8·1	8·2	3·50	2·9	69·90	0·6
$\frac{1}{8}$–$\frac{1}{16}$ . .	11·70	8·14	8·7	4·31	3·4	62·71	0·8
Total .	100·00	8·55	8·5	3·70	1·6	66·40	1·25

The raw coal was separated into 92·85 per cent. of clean coal, with an ash content of 3·70 per cent. and containing an average of 1·6 per cent. of particles sinking at S.G. 1·5, and 7·15 per cent. of refuse with an ash content of 66·40 per cent. and containing 1·25 per cent. of particles floating at S.G. 1·5. The loss of coal, therefore, amounted to 0·1 per cent. of the total coal fed.

The results in Table 96 were obtained with a typical Staffordshire coal, the coal above 1 in. and below $\frac{1}{16}$ in. being untreated.

These results show that, in cleaning these four sizes of coal, the mean ash percentage was reduced from 14·7 per cent. in the raw coal to 2·7 per cent. in the clean coal, with a loss of 0·9 per cent. of coal in the refuse, equivalent to a loss of 0·2 per cent. of the total coal

TABLE 96.—RESULTS OF CLEANING ON S.J. SEPARATOR

Size (in.)	Raw Coal.			Clean Coal.		Refuse.	
	Per cent of Sample Treated.	Ash per cent	Per cent. Sinking at S.G. 1·5	Ash per cent.	Per cent. Sinking at S.G. 1·5	Ash per cent.	Floating at S.G. 1·5
1–½ . .	36·5	14·4	22·0	2·75	0·9	83·1	1·25
½–¼ . .	27·9	12·5	17·2	2·45	1·2	82·5	0·4
¼–⅛ . .	25·1	15·3	20·9	2·75	1·4	82·2	0·7
⅛–1/16 . .	10·5	19·5	26·7	3·15	2·9	81·2	1·25
Total .	100·0	14·7	21·3	2·7	1·3	82·5	0·9

fed. The efficiency of cleaning is, therefore, very high, though it should be remembered, in comparing these results with those of cleaning by other processes, that the coal was carefully screened into four separate sizes before treatment.

A number of other results were given by Appleyard (*Trans.* Inst. Min. Eng., 1927, 73, 404).

The Wye Separator.—The Wye separator, introduced in 1926, is an improvement on the S.J. separator, and has the advantages that it can clean coal below ⅛ in., does not require such close sizing of the feed, and has a higher capacity for the same floor space. In one plant erected by the Birtley Iron Company for a capacity of 80 to 100 tons per hour, the feed coal is sized into two fractions only—1½ to ½ in. and ½ in. to 0.

It differs from the S.J. separator in the shape and arrangement of the deck. The principles of its action are similar, stratification being induced by an air current passing upwards through a perforated surface, aided by the vibration of the deck, but the separation of the stratified layers is rather differently accomplished. In early models the understructure and the actuating mechanism were the same as on the S.J. table, but in 1927 the new Mark IV mechanism was introduced.

The Y-shape of the deck is shown in Figs. 150, 151 and 152. The raw coal is fed into the pan marked "Raw coal feed" (Fig. 151), whence it is precipitated through the central slot on to the deck by the oscillating motion of the table. The deck is concave at the feed end, but rises centrally towards its narrowest width, where it is convex. At the feed end, also, the riffles are arranged (Fig. 152) so that they converge towards the axis. If a line be drawn along the axis, the riffles may be regarded as forming a series of co-axial V's. From the feed to the narrowest part of the deck the apex of

each V is nearer to the refuse discharge end of the table. From the narrowest portion of the table to the refuse and middlings discharge positions at the end of the arms of the Y, the riffles are inclined to the sides of the table and are parallel to the axis.

The coal spreads out across the table at the feed end, the combined action of the shaking motion and the air current resulting in a stratification with the coal in the uppermost layers. The refuse particles in the lower layers are trapped between the riffles; and the motion of the deck causes them to pass forward and to collect along the axis where the riffles converge. The coal, however, is squeezed out as the bed narrows and the deck becomes convex, and is made to travel towards the sides over which it is discharged into the clean coal collector. The separation is shown clearly in Fig. 151.

The primary or rough cleaning of the raw coal is accomplished

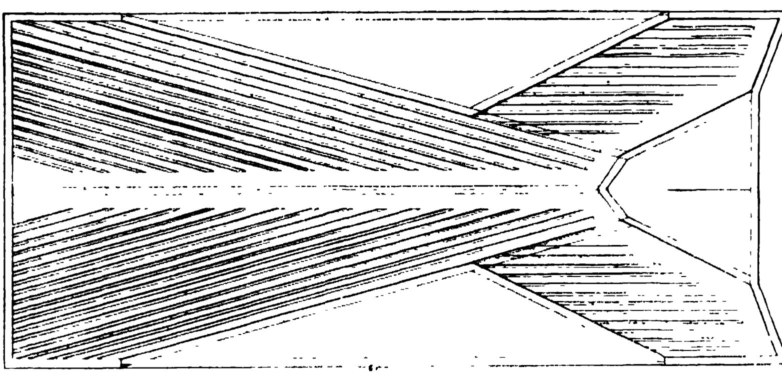

FIG. 152.—Wye Separator. Arrangement of Riffles.

on this first portion of the deck, on which the material travels slightly uphill. On the second portion, in the arms of the Y, the refuse is cleaned to remove coal particles from it. The arms are inclined downwards from the centre to the sides, and along them the riffles run parallel to the axis. The refuse between the riffles is therefore carried to the inside of the arms, where it encounters the banking bar. Here, as on the S.J. separator, it mounts up and forms a solid bank, the coal and middlings being squeezed away from it towards the sides. At the junction of the arms is a central barrier consisting of two tubes connected to the air supply. Air is distributed outwards from them over the refuse which is beginning to pile up against the banking bar and the air current, together with the inclination of the surface, forces the coal away towards the sides.

The clean coal collects along the whole length of the sides of the table, the smallest sizes coming over near to the feed end and the largest particles along the sides of the arms of the Y. The

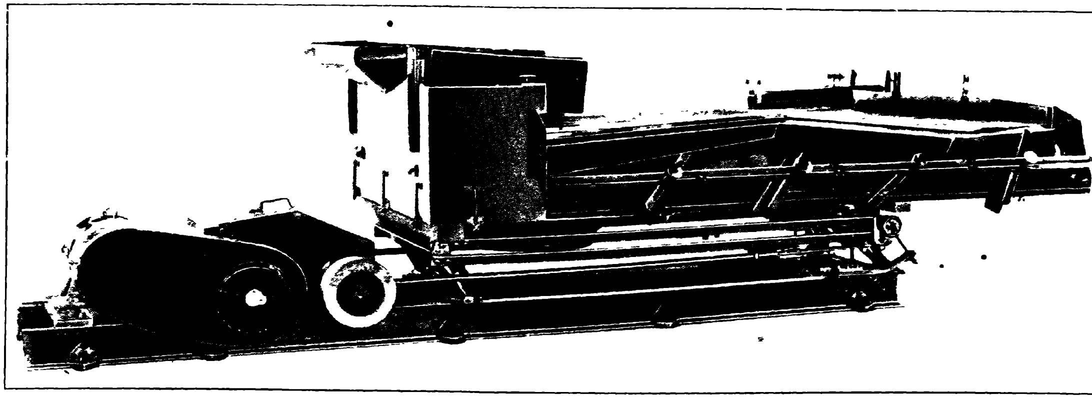

FIG. 150.—The Wye Separator Side View.

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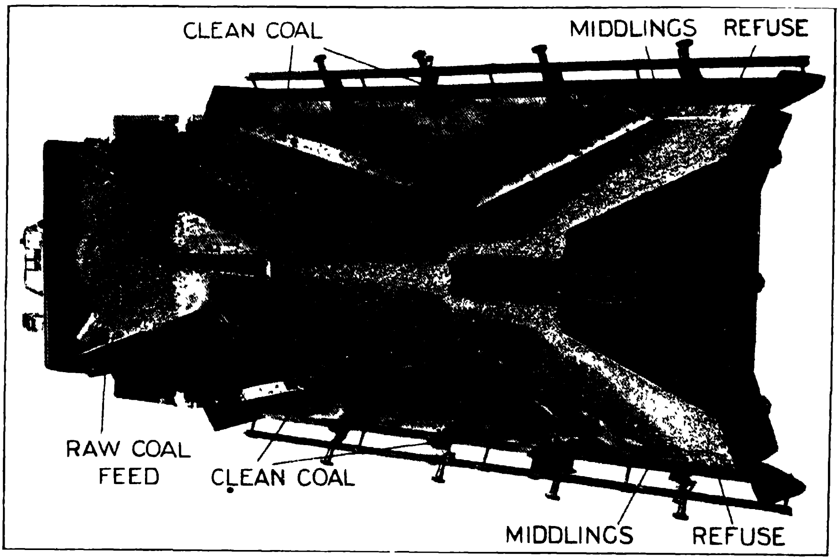

FIG. 151.—Separation of Coal and Dirt on Wye Separator.

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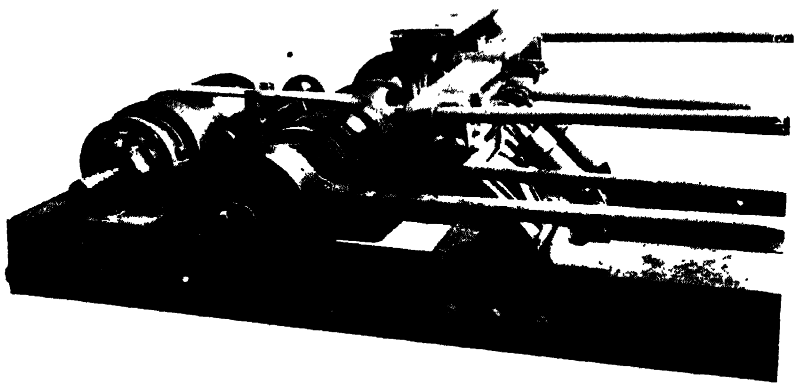

Fig. 153 — Wye Separator. Driving Mechanism.

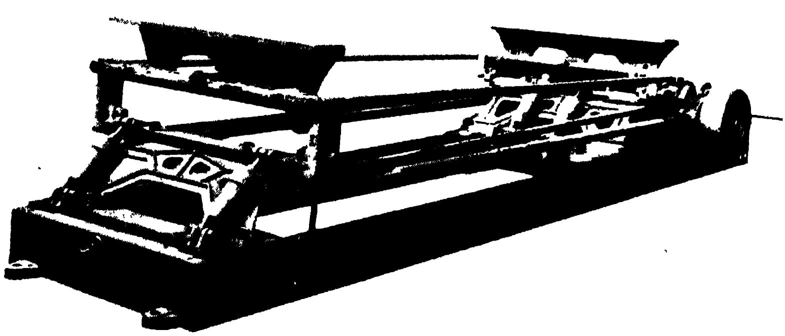

Fig. 154 — Wye Separator. Understructure.

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refuse and middlings collect at the end of the arms, and dividing cutters enable the products to be collected separately. It will be observed that the discharge of the clean coal is distributed over a greater lateral distance than was the case on the S.J. separator, and it is therefore easier to collect portions of the product of different qualities. The middlings and refuse are, however, congested, as on the S.J., and the division between them, and also between the middlings and clean coal, is not so definite as on a wet concentrating able. This is corrected by re-circulating the middlings fraction.

The deck of the Wye separator is of similar construction to that of the S.J. (see Fig. 149). For coal below $\frac{1}{8}$ in. in size it consists of a perforated zinc plate instead of a wire-mesh gauze, and the plate is slightly corrugated. For material above $\frac{1}{8}$ in. bronze wire-mesh gauze is used and the corrugations are omitted. The surface is supported on longitudinal wooden strips, arranged parallel to the riffles on the deck, and the riffles are tacked to the strips. The air supplied under the table is distributed by a perforated vane with adjustable flaps.

For coal below $\frac{1}{8}$ in., the riffles, beginning from the feed end of the table, are $\frac{1}{4}$ in. high on the axis of the table, and increase in height to $\frac{3}{4}$ in. at the edge. They taper from the feed end towards the narrow part of the table, where they are all $\frac{3}{32}$ in. high. There are fourteen riffles on each side of the deck. Along the arms, where the riffles are parallel to the axis, the heights vary from $\frac{1}{4}$ to $\frac{3}{4}$ in. and taper to a height of $\frac{3}{32}$ in. near the centre to $\frac{1}{32}$ in. at the finishing end. There are thirty-four riffles on each arm. This is one arrangement of riffles. Different coals require different arrangements to obtain the best results, and modified riffling is employed where necessary.

The Mark IV actuating mechanism and understructure for the Wye separator is shown in Figs. 153 and 154. In principle it is the same as the mechanism fitted to the S.J. table, but it is more rigidly constructed. The support for the deck framework consists, as before, of two inclined rocking plates bedded in V-shaped grooves. This is plainly visible in Fig. 154. The springs have been replaced by others of the barrel type, which are stronger and more easily take up play in the rocker plate bearings. The springs are enclosed in casings. The methods of obtaining the side and end inclinations have been modified; the slots are retained, but the deck is locked firmly by fixed bolts instead of by the handwheel device on the S.J. table, the locking being thereby strengthened. The adjustment of the inclination is thus rather less accessible.

In Fig. 153 the variable speed cone pulleys for driving the eccentrics can be seen. The handwheel and screw adjustment for the belt enables the speed of the drive to be varied easily and whilst the table is running. The eccentric drive is transferred by the connecting rod to the distant, or refuse, end of the table directly, the rod being now solid with a flattened end instead of (as previously)

fitted with a screwed joint. At the refuse end the connecting rod is fastened directly to the cross-piece, resting on the top of the inclined rocker plate.

In operation, the Wye separator can be used for coal below 3 in. and greater than $\frac{1}{64}$ in. With coal from 2 to $\frac{1}{64}$ in. the feed should be sized before cleaning into at least three sizes, 2 to $\frac{1}{2}$ in., $\frac{1}{2}$ to $\frac{1}{8}$ in., and $\frac{1}{8}$ to $\frac{1}{64}$ in. Coal smaller than $\frac{1}{64}$ in. cannot be properly treated because it remains supported in the air currents and complicates the operation. It is therefore removed by aspiration before the coal is fed to the table. The capacity of the separator is given in Table 97.

TABLE 97.—WYE SEPARATOR. CAPACITY AND POWER REQUIREMENTS

Size of Coal (in.).	Capacity. Tons per hour.	Power Required. H p.
$2-\frac{1}{2}$	60–70	30
$\frac{1}{2}-\frac{1}{8}$	30–40	16
$\frac{1}{8}-\frac{1}{64}$	20–25	7

The actual capacity and the most desirable sizing limits depend upon the coal treated. The usual sizes are 2 in., $\frac{1}{2}$ in., and $\frac{1}{8}$ in., but in one plant recently erected the coal is sized into fractions $2\frac{1}{2}$ to $\frac{7}{8}$ in., $\frac{7}{8}$ to $\frac{1}{4}$ in., $\frac{1}{4}$ to $\frac{1}{12}$ in., $\frac{1}{12}$ in. to 0.

For small collieries, where lower capacities are required, a

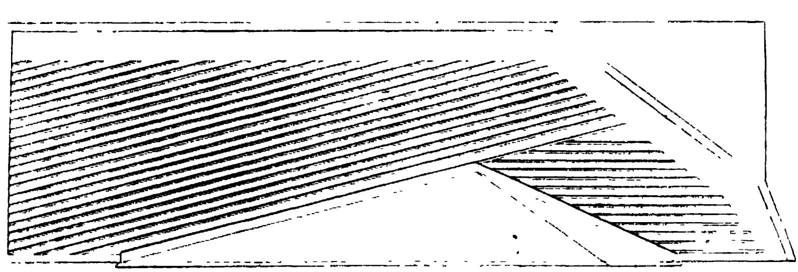

FIG. 155.—Half-Wye Separator: Arrangement of Riffles.

Half-Wye table is supplied. A plan of the riffling of the deck is shown in Fig. 155. The coal is fed at the left-hand end; clean coal discharges along the bottom edge of the table and the refuse at the right-hand end.

The Half-Wye table consists of a Wye table split along the axis. It is especially suitable for collieries with small outputs. In such a case it might occur that sufficient of the $\frac{1}{2}$ to $\frac{1}{8}$ in. and $\frac{1}{8}$ to $\frac{1}{64}$ in.

fractions were present in the raw coal to utilise the full capacity of a Wye separator, but that only a small proportion of 2 to $\frac{1}{2}$ in. coal was present. A Half-Wye separator could then be used for this fraction.

Before the coal is fed to Wye separators the bulk of the fine dust is removed by aspiration. The aspirator, shown in Fig. 156, is a succession of cascades over which the raw coal falls and the dust is removed by a current of air. Appleyard (*loc. cit.*) gives the sizes of the feed to the aspirator and the products obtained, as in Table 98.

TABLE 98.—SIZES OF COAL AND DUST OBTAINED BY ASPIRATION

Size (in.)	Feed to Aspirator per cent.	Coal Discharged per cent.	Dust Obtained per cent.
$> \frac{1}{16}$	39·1	48·7	0·5
$\frac{1}{16}-\frac{1}{32}$	27·3	30·2	4·8
$\frac{1}{32}-\frac{1}{64}$	15·3	13·7	15·4
$\frac{1}{64}-\frac{1}{81}$	5·4	3·3	14·8
$\frac{1}{81}-\frac{1}{125}$	4·7	1·6	25·0
$< \frac{1}{125}$	8·2	2·5	39·5
	100·0	100·0	100·0

In this case, the raw coal had an ash content of 12·6 per cent., but the aspirator dust had an ash content of only 8·3 per cent.

The dust passing forward in the air current is passed through a series of low-pressure bag-filters and recovered for use as pulverised fuel or for mixing with the coking slack. The disposal of this dust, however, is a matter of some difficulty. A colliery company may be loathe to replace its existing boiler plant by pulverised fuel equipment, and finely-powdered coal dust is regarded as a dangerous material when transported by rail. It requires special packing, or special types of wagons, and an increased freight rate is charged.

One other difficulty in the operation of a pneumatic cleaning plant is the behaviour with a feed varying in quality and quantity. To overcome the difficulty of an irregular rate of feed, adequate storage capacity must be provided for each table, and the table must be stopped if the hopper empties sufficiently to reduce the feeding rate. When the proportions of coal and shale in the feed are apt to vary, it is necessary to collect a larger proportion of middlings for re-circulation. With a reduction in the proportion of dirt, some coal may be caught in the riffles and travel too far along the surface of the table towards the refuse-discharge point. On the other hand, with an increased proportion of dirt, some of the heavy particles may

be squeezed out of the riffles and there may be difficulty in removing them completely from the coal.

When a pneumatic separator is started up, normal conditions are not experienced for some minutes, and inefficient separation obtains. It therefore follows that troubles will arise if the feed is intermittent and it is frequently necessary to stop and start the plant.

Another difficulty to which all dry-cleaning processes are subject is caused by moisture in the coal. The fixed moisture retained on air-drying is of no consequence, but the adherent moisture, if it exceeds 2 or 3 per cent., may lead to difficulty of operation.

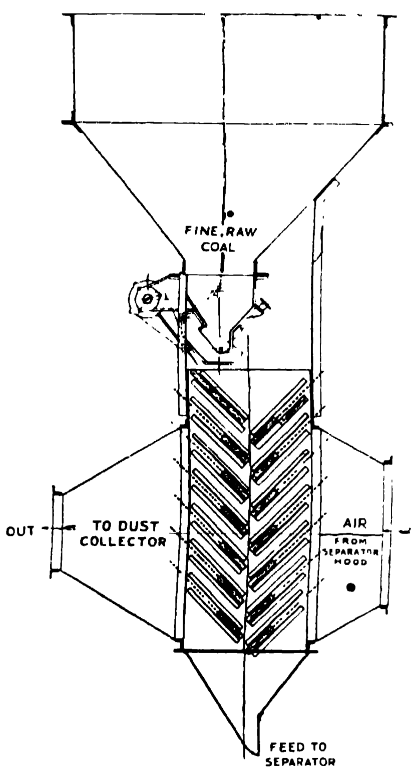

Fig. 156.—Section through Dust-Extraction Device.

The Wye separator has not yet been in commercial operation in this country for a sufficient length of time to prove its ability to meet British conditions. In America five tables have been in operation for about two years, and the average results during that period show that the ash of a West Virginian coal has been reduced from 12·62 to 7·09 per cent., the refuse containing 81·12 per cent. of ash. Unfortunately, average results of float and sink tests are not available. The ash content of the coal floating at S.G. 1·5 was 5·45 per cent., so that probably a quantity of high ash material must have been included in the clean coal. The refuse had a high ash content, and the figures suggest that the coal was easy to clean.

On the $\frac{1}{2}$ to $\frac{1}{8}$ in. fraction of a South Yorkshire coal, test figures show that 13·7 per cent. of the feed sank at 1·5 S.G. The clean coal contained 1·35 per cent. of sinks in 1·5, and the refuse 1·95 per cent. of floats. The ash content was reduced from 11·72 to 2·90 per cent., the refuse containing 70·0 per cent. of ash.

The first important plant in this country to use pneumatic tables for coal cleaning was at Wardley, County Durham, where the coal below 2 in. from the Follonsby and Springwell collieries is treated. The plant has a capacity of 125 tons per hour. The

flow-sheet is given in Fig. 157. The raw coal is shot into a 25-ton storage hopper, whence it is elevated to the top of the cleaning building and discharged into a worm conveyor and on to a $\frac{1}{2}$ in. mesh vibrating wire screen. The oversize from the $\frac{1}{2}$ in. screen is

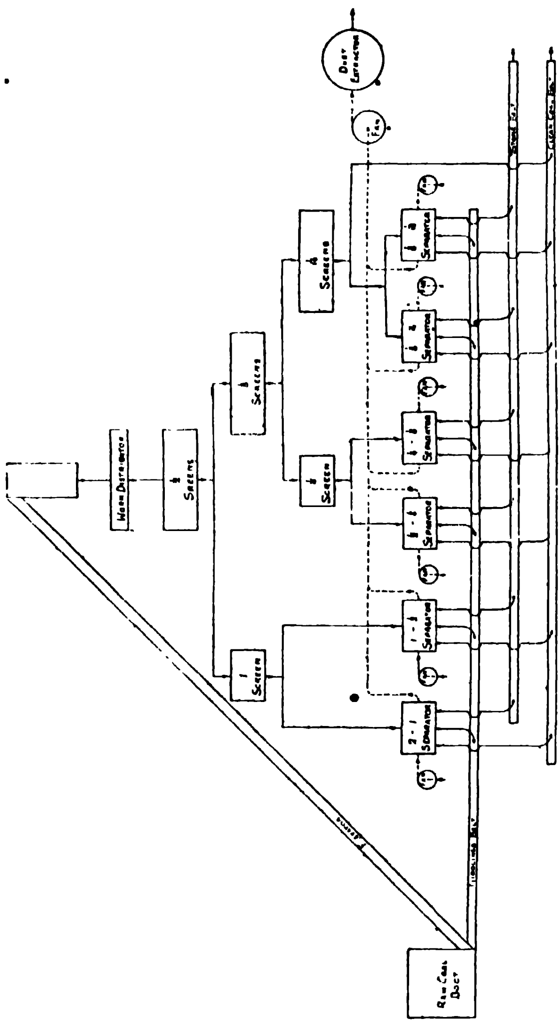

FIG. 157.—Flow Sheet Wardley Pneumatic Coal-Cleaning Plant.

passed to a 1 in. screen and divided into fractions of 2 to 1 in. and 1 to $\frac{1}{2}$ in. The undersize is divided on a $\frac{1}{8}$ in. screen, the material passing through a $\frac{1}{8}$ mesh being subsequently divided into $\frac{1}{8}$ to $\frac{1}{16}$ in., and $\frac{1}{16}$ in. to 0, the latter portion being by-passed. The $\frac{1}{2}$ to $\frac{1}{8}$ in. size is divided into two further fractions by a $\frac{1}{4}$ in. screen.

The cleaning plant consists of six S.J. tables, one for each of the four largest sizes and two for the fraction of $\frac{1}{8}$ to $\frac{1}{16}$ in. The coal is

fed to the tables by shovel feeders bolted to the bottoms of storage hoppers. The six separators are all situated on one floor of the building. Each separator is provided with its own fan and motor, the motor driving the fan, the reciprocating mechanism and the shovel feed. Each table is, therefore, a separate unit and separately controlled. Separate control, whilst desirable from the point of view of efficient operation, especially on a coal consisting of a mixture of two raw coals which may be delivered irregularly, is expensive, both in power, upkeep and supervision. In America a plant has been tried without a separate fan and driving motor for each table, but the design was not a success, and such an arrangement is not considered suitable for future plants.

Each table and each of the screens is covered by a dust extractor hood and duct, the sides of the tables being enclosed by canvas. The agitation of coal must, of necessity, result in some fracture of the coal, and the dust produced is removed in the air current. In later plants suction is applied to the collecting hoods. The dust-laden air is passed into an extractor house, where it is filtered by special fabric bags.

The products from the six separators are discharged down separate shoots, through the second floor, on which the fans and motors are housed, to the first floor, where they are delivered on to travelling rubber belts. The two clean coal belts are totally enclosed and connected with the dust-extraction system. On them the various sizes of coal are remixed. A third rubber belt conveys the refuse out of the building, and a fourth delivers the middlings product to the foot of the raw coal elevator for re-cleaning.

The air requirements for the Wardley plant are given in Table 99.

TABLE 99.—OPERATING DATA. S.J. SEPARATORS. WARDLEY

Size Treated (in.).	Cubic feet Air per minute.	Water Gauge (in.).	Capacity Tons per hour.	H.p.
2-1	17,000	5·5	30	29
$1-\frac{1}{2}$	12,000	5·0	30	18
$\frac{1}{2}-\frac{1}{4}$	8,500	5·0	21	16
$\frac{1}{4}-\frac{1}{8}$	6,300	3·5	17	9
$\frac{1}{8}-\frac{1}{16}$	5,000	3·0	12	7

The Wardley plant was in operation for a few months before the coal stoppage of 1926, and has been in continuous operation since then with satisfactory results.

The Arms Air Concentrator.—The Arms air table, shown in Fig. 158, is a pneumatic separator depending for its action upon

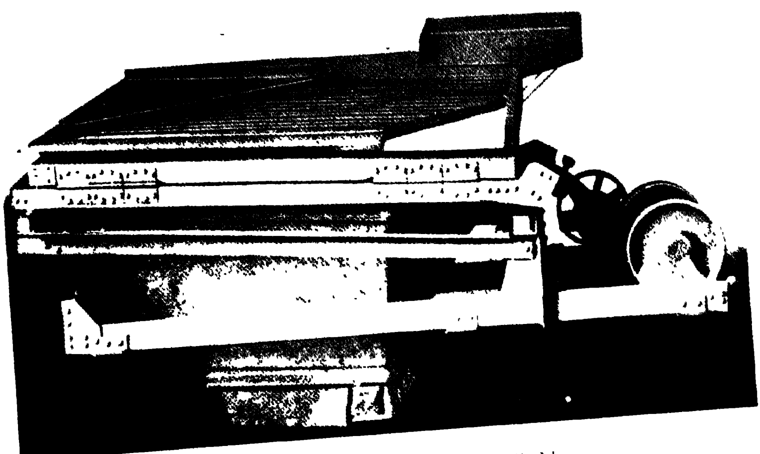

FIG. 158. The Arms Pneumatic Table

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exactly the same principles as those of other pneumatic tables. It was designed by the Roberts and Schaefer Company, of Chicago, U.S.A., and was erected in this country by Messrs. Hugh Wood & Co., Ltd. In shape and general design it is similar to the S.J. table.

The concentrator consists of a vibrating surface with a perforated mesh top through which a current of air is blown from a fan situated beneath it. The deck is covered with overlaid chicken-wire and metal riffles which guide the refuse towards its discharge point. For coal of size from $\frac{3}{8}$ to $\frac{3}{16}$ in. there are about thirty riffles tapering from a height of 1 in. at the mechanism end to $\frac{1}{4}$ in. at the refuse end of the table, but different sizes and dispositions of the riffles are used for different coals. The riffling shown in the photograph (Fig. 158) is parallel to the discharge side of the table. Frequently better results are obtained when the riffles are inclined to the side, and this has been found desirable in the majority of plants erected.

The coal is fed at the highest point of the table (in Fig. 158 at the top right-hand corner) and stratifies as a result of the air current through the bed and the shaking motion of the deck. The clean coal in the upper layers passes straight down the table to the discharge side, the refuse being carried across the table between the riffles. The shape of the table is more nearly triangular than that of the S.J. separator. The barrier across the head end of the coal discharge side (the bottom right-hand corner in Fig. 158) is sometimes omitted and the barrier at the opposite end of the discharge side (the bottom left-hand corner in Fig. 158) is carried diagonally across the table and is not " dog-legged " as on the S.J. separator. The effect of this is that the refuse does not bank up against the barrier, but is spread out over a wider lateral area. This arrangement probably reduces the amount of middlings, because the products are discharged over a greater length and the division between clean coal and middlings and between middlings and refuse can be more carefully and exactly set. Consequently, it may not be necessary, on the Arms table, to re-circulate such a large quantity of middlings as on the S.J. table. On the other hand, the banking of the refuse against the barrier, which is one of the features of the design of the S.J. and Wye separators, does not occur, and the probability of middlings and of clean coal particles being lost in the refuse may be increased.

In Fig. 158 the adjustable cutter plates which cause the products to fall into separate compartments have been omitted. As a rule the clean coal and middlings are collected along the near side of the table and the refuse at or round the left-hand bottom corner.

The inclination of the table deck is adjustable from the feed side to the discharge side and from the mechanism end to the refuse end. The highest point of the table is the corner where the feed is supplied and the lowest is the corner between the discharge side and the mechanism end. At this corner the smallest clean coal collects. The table is inclined upwards from the mechanism end

to the refuse end, so that the refuse passes uphill and any coal particles entangled with it are no longer carried forward in the dirt if they can rise to the uppermost layers. The table is hinged along the side (Fig. 158) to provide the slope down the table. The slope is adjusted by two pins at the back of the table sliding in curved slots to which they can be locked in any desired position. The inclination from end to end of the table is adjusted by a screwed rod working in a thread fixed to the main steel base.

The table is supported on a base of steel channels provided with cross-pieces. The reciprocating motion is obtained by an eccentric connected to a driving rod. The eccentric is driven by a pulley operating through a Reeves variable-speed transmission. The Reeves gear consists of two pulleys keyed to parallel shafts, each pulley consisting of two cone shape discs with their apices facing. One disc is fixed to the shaft, the other can slide along it, so causing the belt to drive nearer to or further from the axis.

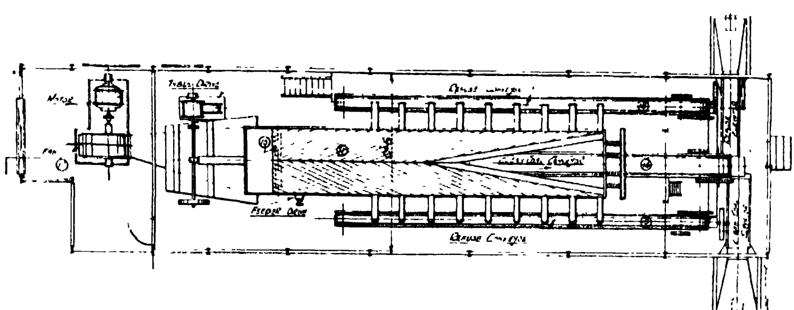

FIG. 159.—The Peale-Davis Table. Plan.

The sliding halves of each pulley are connected so that one adjustment automatically regulates the distance between the faces of each pulley. By this means the gear ratio may be changed between limits. The gear is similar to the old Rudge motor-cycle multi gear and the Zenith gradua gear.

The framework of the deck, to which the reciprocating motion is transmitted, is supported on an underframe by two short rocking arms. The underframe is fixed rigidly to the main steel base channels.

In America there are fifteen plants using Arms tables with a combined hourly capacity of 1,870 tons of coal per hour. At the Ashington Colliery, Northumberland, England, two Arms tables treat 40 tons of coal per hour. Before cleaning, the coal is sized into fractions with a size ratio of 2 : 1. The screen used is the Arms ccreen, consisting of a wire-mesh surface, placed nearly horizontally, and vibrating rapidly to and fro.

The Peale-Davis Table.—The Peale-Davis table was first erected in America in 1924, and the first plant in England is being

installed by the Woodall-Duckham Company for the Nunnery Colliery Co., Ltd., at Handsworth, near Sheffield, in connection with the Becker coke ovens which they are also supplying.

The table differs from other pneumatic tables in that it has a higher capacity and treats unsized coal. The principles of its action are, however, similar, a current of air passing upwards through a bed of material on a perforated and riffled deck, the table being oscillated in a longitudinal direction. It is made with capacities from 25 to 250 tons per hour, and the raw coal may consist of all sizes from 3 in. to 0.

The table is shown in plan in Fig. 159. For a capacity of

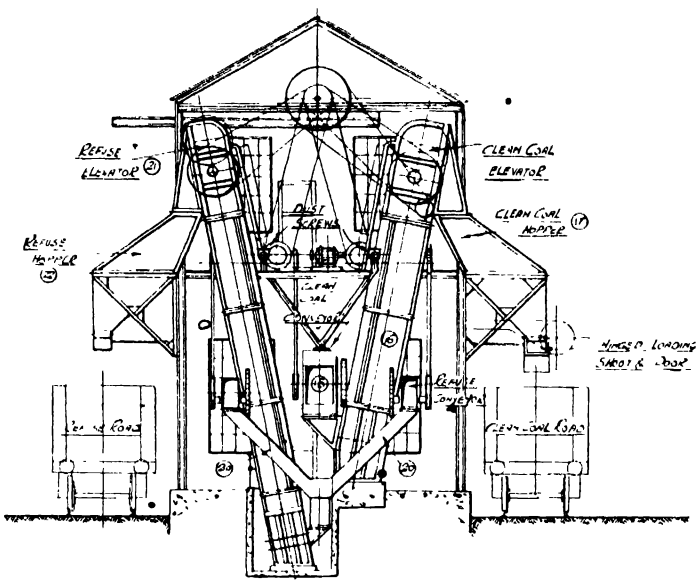

FIG. 160.—Section through Plant using Peale-Davis Table.

250 tons per hour, the deck, 11, is 40 ft. long and 14 ft. wide. The riffles are disposed in a series of V-shapes, as shown, and taper from the axis towards the sides. The deck consists of perforated plates, sloping upwards from the axis, and the perforations are spaced so that they are closer together near the feed end of the deck. By this means the air current is directed to the parts where the bed is thickest.

The raw coal is supplied from a hopper, through feeders, 10, which ensure a regular rate of feed. The dirt, which sinks to the bottom of the bed, is trapped between the riffles and passes outwards from the centre to the sides where it is collected in a trough and discharged through shoots to two belt conveyors, 18 and 19. The coal, however, passes over the riffles and towards the axis of the table and is finally discharged on to the clean coal conveyor, 14,

running beneath the table and parallel to the axis. The coal and refuse are elevated and disposed of as required.

The table is made to vibrate by an eccentric mechanism driving against springs at the discharge end, thus ensuring a rapid reversal from the forward to the backward stroke. The deck is supported on short rocking arms which are inclined to the vertical so that, on the forward stroke, the deck rises slightly and falls on the backward stroke. The deck is inclined longitudinally, the feed end being lower than the discharge end.

The air is supplied continuously by a fan, 12, to an air chamber below the deck. The amount of air varies according to circumstances, but for large capacities is about 600 cub. ft. per minute for every ton of hourly capacity (30,000 to 40,000 cub. ft. per ton treated). In operation, the air blows some of the dust from the coal, and this is collected by means of an expansion chamber provided with dust pockets, no fans, cyclones or filters being utilised.

A section through the plant is shown in Fig. 160. The shoots, 20, deliver the refuse to the boot of the elevator, 21, which discharges it into the hopper, 22. The clean coal passes from the end of the conveyor, 15, to the foot of the elevator, 16, and into the hopper, 17 The clean coal and refuse may then be delivered to railway wagons or disposed of by other means.

No results are, of course, available of operation on British coals, but tests on Pennsylvania coals gave the results in Table 100. The raw coal was 3 in. to 0 in size.

TABLE 100.—RESULTS OF OPERATION—PEALE-DAVIS TABLE

Raw coal	10·5
Clean coal . . .	7·5
Refuse	50·5

Float and sink tests on the raw coal gave the results in Table 101.

TABLE 101.—FLOAT AND SINK RESULTS, PENNSYLVANIA COAL

S G.	Wt. per cent. of Total	Ash Content per cent.	Wt per cent. Cumulative	Cumulative Ash Content per cent.
< 1·3 . .	53·0	3·2	53·0	3·2
1·3–1·4 . .	25·8	7·5	78·8	4·6
1·4–1·5 . .	6·7	13·5	85·5	5·3
1·5–1·6 . .	6·5	20·9	92·0	6·4
1·6–1·75 . .	2·0	29·9	94·0	6·9
> 1·75 . .	6·0	66·9	100·0	10·5

From these results, it will be seen that the coal was not easy to clean, for it contained a relatively large proportion of middlings. Moreover, the coal fed contained 27·6 per cent. less than $\frac{1}{8}$ in. and 47·6 per cent. between $\frac{3}{4}$ and $\frac{1}{8}$ in., the remaining 24·8 per cent. being up to 3 in. in size. In these circumstances the results were quite satisfactory.

The purpose of the colliery was to produce a clean coal containing less than 8 per cent. ash. The clean coal actually delivered comprised 93 per cent. of the raw coal with an ash content of 7·5 per cent., which is within 0·75 per cent. of the theoretical minimum at a yield of 93 per cent.

The possibility of treating unsized coal, with only one table for a large capacity, suggests that the Peale-Davis table may be preferable to other types, and operating results on British coals will be watched with great interest.

The Raw Process.—The Raw process has been in operation at the Murton Colliery of the South Hetton Coal Co., Ltd., Co. Durham, for about twelve months, and more recently at several other collieries in the county of Durham. It differs from all other processes in that a vibratory motion of the separating surface is combined with a pulsatory air pressure.

The coal is fed to the upper end of a longitudinally inclined table, from a device for ensuring a definite rate of feed, and travels forwards along the table in the direction of the slope. The table is actuated by eccentrics, the vibratory motion being parallel to the longitudinal axis. A stationary air chamber below the separating surface is connected thereto by flexible material and a fan capable of generating a suitable pressure of air communicates with the air chamber through a pulsator. The pulsator consists of a shaft carrying a pair of discs or spiders to which two oppositely-disposed circumferential vanes are attached. As the discs revolve, the vanes rapidly pass a port in the end of the air chamber and sharp periodical fluctuations of pressure in the air chamber are thus produced.

In operation, unsized coal below about 2 in. is fed to the table, and a layer of about 4 to 6 in. deep travels down the deck. The fluctuating static air pressure, produced by the operation of the fan and the pulsator, supports the bed *en bloc*, with a minimum volume of air leaking through. In the quasi-suspended condition thus produced, the denser material can fall to the lowest layers of the bed, displacing the lighter materials bodily. It is said to be a true density differentiation, independent of the influence of size.

In plan, the table is divided into three portions, each consisting of a rectangular trough or cell. The troughs are of different widths, the one at the feed end being the widest and the others progressively narrower. The floor of each trough is set at a slightly lower level than the preceding one, and they are connected by short relatively steep steps. Above each step is a skimmer, consisting of a horizontal

sheet of perforated metal with vertical walls in the shape of a V, whose point is towards the feed end of the table. These skimmers guide the upper layers of less dense material which have been stratified in the preceding cell to the sides of the table to be discharged. The object of the perforated base of the skimmers is to allow the removal of the upper layer to take place under the same static pressure conditions as obtained in the stratifying cell, thereby avoiding remixing of the stratified materials. The sides of the table are slotted at the forward end of each cell, to allow the skimmers to be placed at a suitable level in the bed, and the skimmers converge towards the axis of the succeeding cell. The lower, and so far incompletely stratified layers flow down the step from the first cell into the next cell, which, being narrower, accommodates the material at the same depth as in the preceding cell. Further stratification is here achieved, and the remainder of the clean coal is removed by a second skimmer, below which the refuse and middlings flow down the third step into the last and narrowest cell. In this cell, the middlings and the refuse are separated, the middlings being removed by a skimmer. The refuse flows over a shoot which is attached to the extreme forward end of the deck and whose inclination is adjustable. By adjusting the upward inclination of the shoot relative to the deck, the amount of material discharged as refuse is under control. This final separation is under visual observation and responds to immediate adjustment.

The appliance may be used with a steady, instead of a fluctuating, air pressure, but the pulsations maintain a more mobile condition in the bed, and thus facilitate separation.

These tables are capable of handling 50 tons per hour in one size of unit, and 25 tons per hour in the case of a smaller unit.

CHAPTER XIX

MISCELLANEOUS DRY-CLEANING PROCESSES

A NUMBER of attempts have been made to separate coal from dirt by taking advantage of their different coefficients of friction. The amount of friction between a particle of shale and a plane over which it slides is greater than the friction between the same plane and a particle of coal of equal weight. If, therefore, particles of each material be given a certain impetus and slide over a surface for a given distance, their resultant velocities are different, and means may therefore be devised by which the particles can be separated. The difference between different methods of effecting the separation of coal from dirt by taking advantage of their different coefficients of friction lies in the manner in which the different velocities are utilised.

In practice, when the separation is effected on an inclined plane, the particles are shot on to the plane and they therefore begin to slide with an equal initial velocity (u). The forces acting upon any given particle are, firstly, the component of the force of gravity resolved down the plane; and, secondly, the frictional resistance to motion. The resultant acceleration of the particles sliding down the plane in air may therefore be expressed by the equation

$$\frac{ar^3s}{g}\frac{dv}{dt} = ar^3s \sin\alpha - ar^3s\,\mu\cos\alpha$$

adopting our standard notation. The acceleration, $g(\sin\alpha - \mu\cos\alpha)$ is constant, involving only terms which are constant for a given particle.

After sliding a distance, l, the velocity of the particle will be

$$v = \sqrt{u^2 + 2gl(\sin\alpha - \mu\cos\alpha)}.$$

If the plane is l units long, when the particle reaches the end of the plane and slides off it, it will be moving with this velocity in a direction inclined at an angle α to the horizontal. This velocity may be resolved into two components, equal to $v \sin\alpha$ in a vertical direction, and $v\cos\alpha$ in a horizontal direction. Its trajectory will, therefore, be the result of a vertical acceleration due to gravity from an initial velocity of $v \sin\alpha$, and a constant horizontal velocity equal to $v\cos\alpha$.

Assuming the fall to be arrested by the particle falling on to a plate placed x units of length below the lip of the plane, the time to reach the plate will be

$$\frac{1}{g}(\sqrt{v^2 \sin^2 \alpha + 2gx} - v \sin \alpha).$$

During this time the particle will have travelled a distance equal to

$$\frac{1}{g} v \cos \alpha \, (\sqrt{v^2 \sin^2 \alpha + 2gx} - v \sin \alpha)$$

in a horizontal direction.

For two particles, one of coal and one of shale, the distance travelled horizontally will be different during the time required to reach the plate, because the value of v will be different. They will therefore reach the plate at different points. By taking advantage of this fact, the coal can be made to fall on one side of a partition and the shale on the other side of it.

In practice the interference created by a number of particles moving together results in any such simple method of collection being inadequate, though it has been used with fairly satisfactory results for special work both in ore-dressing and coal-cleaning practice. The commonest way in which it has been employed, however, and that only rarely, was by an arrangement of several inclined planes, spaced laterally with gaps between them. The coal particles could jump across the gap, whereas the shale particles could not do so, but fell through the gaps into a collector placed beneath them.

Several other modifications of the method have been employed from time to time, but, except for the simplest purposes and for relatively large particles (say, over $\frac{3}{4}$ in.), they have never come into general use.

Nevertheless, there are other appliances which have been employed extensively for the separation of dirt from coal and which depend upon similar principles. Certain revolving tables and spiral separators are cases in point. In each process the coal and shale are made to take separate paths whilst still in contact with the separating surface, the paths being the result of different amounts of frictional resistance to their motion. Before describing them, certain inherent drawbacks to processes dependent upon frictional differences may be mentioned.

There are a number of factors which influence the velocities acquired by sliding particles and which, therefore, have a bearing on the design of apparatus utilising the property of friction. In general, these factors, such as the shapes of the particles, the exact hygroscopic state of their surfaces and of the surface of the plane, vary from coal to coal, and make the adoption of any standard design of limited application. The object of the design is to cause the particles to acquire as widely differing velocities as possible. For example, on an inclined plane, the best inclination would be one which caused an acceleration of the coal and a retardation of the dirt. For such a design to be of wide application and to apply it extensively, it would be necessary that different coals should vary

little in certain properties. On the other hand (Table 102), it has been found that different coals have different coefficients of friction, and that variations are also found in the same seam. Similarly there is no common coefficient of friction for the dirt associated with coal, the variation being greater than for coal particles because the nature of the dirt varies in different seams, and even in different partings of the same seam.

Frictional resistance is also of considerable importance in the use of inclined screens, and it is convenient to state here some of the experimental results that have been obtained, and which are applicable to both coal-cleaning and screening appliances. The coefficient of friction of a particle on a given surface can be measured by the angle at which the surface must be inclined for the particle to begin to slide down the plane. This angle determines the coefficient of "static" friction. In practice the coefficient required is that for "kinetic" friction, and this is measured by the angle of the plane at which the particle, if it is caused to move, will continue to move. A number of experiments have been conducted to determine this angle for different coals and other minerals, a few selected results being quoted in Table 102.

TABLE 102.—DATA RELATIVE TO FRICTION

Substance	Nature of plane	Angle of slope to maintain motion Degrees	Authority.
Coal (large lumps) . .	Steel plate.	26	Louis.*
Quartz	do.	25½	do.
Limestone . . .	do.	31½	do.
Anthracite (3¼–2 1/16 in.) .	Glass	10½	Coal Miner's Pocket Book.
do.	Manganese bronze	12½	do.
do.	Sheet iron	14	do.
do.	Cast iron	16	do.
S. Illinois coal			
6-in. lump . .	Bright steel	20½	Holbrook and Frazer.†
Slack . . .	do.	22½	do.
British Bituminous Coal—			
¼ in.	do.	21	Carson.‡
⅛ in. . . .	do.	20½	do.
British Cannel Coal			
⅛ in.	do.	21	do.
½ in.	do.	20½	do.
British Anthracite—¼ in. .	do.	19	do.
⅝ in. .	do.	15½	do.
Charcoal—¼ in. . .	do	13	do.
Curly Shale—1/16 in. . .	do.	18	do.
¼ in. . .	do.	23	do.

* "The Dressing of Minerals," 1909, p. 13.
† Bur. of Mines, *Bull.* 234, 1925, p. 43.
‡ N. of Eng. Inst. Min. Eng., 1927.

There is a third coefficient of friction which may be of considerable importance in certain coal-cleaning appliances and in the design of belt conveyors, namely, the coefficient of rolling friction, but this subject has received little study, and it is usually considered that coal will roll down a slope inclined at 20 degrees to the horizontal, provided that the surface on which the coal rests is in motion.

It will be seen from Table 102 that for lump bituminous coal the sliding angle may vary from $20\frac{1}{2}$ to 26 degrees. The range would probably be found to be still greater if more results were available. For different sizes of a given coal there is a range of several degrees. These differences may be due to the influence of the shape of the pieces. Certain gas coals, which have an approximately cubical shape, will require a less steep inclination than will other coals which break into flatter fragments. In general, shale has a flat shape, the length and breadth of the fragments being greater than the thickness. An occasional thin flat piece of coal, mixed with more nearly cubical particles, might easily, therefore, be lost with the dirt.

These various factors not only influence the design of appliances dependent for their action upon frictional forces, but they are responsible for the fact that no such appliance can be universally applicable to coal cleaning. Certain coals could not possibly be cleaned by such processes because the shale and the coal have approximately the same coefficients of friction. Other coals require very close sizing before they can be cleaned.

It will be seen that, usually, there is a wider difference between the coefficients of friction of anthracite and shale than between bituminous coal and shale, and for this reason anthracite is usually easier to clean on " frictional " appliances than bituminous coal.

The experiments of Nettleton (*Trans.* Inst. Min. Eng., 1921–2,

TABLE 103.—DATA RELATIVE TO FRICTION

	Angle for 80 per cent. to Slide. Deg.
Typical coal-measure shale (Yorks.)—	
$\frac{3}{8}$ – $\frac{5}{8}$ in.	21
$\frac{3}{16}$ – $\frac{3}{8}$,,	23
$\frac{1}{8}$ – $\frac{3}{16}$,,	$27\frac{1}{2}$
Fresh coal	27
Coal stored for four months	$22\frac{1}{2}$
Coal from Silkstone seam (Yorks.)	22
Shale from roof of Silkstone seam	$24\frac{1}{2}$

FIG 161 —View of Spiral Separators in Course of Erection.

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62, 69) are also interesting. Nettleton determined the angles of slope of a sheet of plate glass required in order that mineral matter placed thereon should slide when the surface was tapped. It should be noted that the angles, therefore, relate to static friction. Some of his results are given in Table 103.

From these results the influence of size on the coefficient of friction is apparent; the shale samples between the sizes of $\frac{1}{8}$ and $\frac{5}{8}$ in. required slopes differing by over 30 per cent. For the Silkstone coal and roof shale there is only a difference of $2\frac{1}{2}$ degrees, corresponding to a difference of 12 per cent. in their coefficients of friction. For the sample of coal from the Silkstone seam (Yorks.) the coefficient of friction is 0·4040, for the shale, 0·4557.

The Spiral Separator.—The spiral separator was invented by Pardee and first installed in the anthracite fields of the U.S.A. in 1898, though somewhat similar appliances had previously been used. At one time it was regarded favourably in the U.S.A., and large numbers of separators were in operation in the anthracite districts. A certain number were also employed for the cleaning of bituminous coal, and it is stated that in 1922 there were 4,000 spirals working. In this country spiral separators have been tried at several collieries, and in many instances the operators express dissatisfaction with them; at others they are considered to combine an adequate efficiency of operation with low initial and upkeep costs.

The principal objections raised against spiral separators are:—

1. That they are inefficient after standing for a day or two.
2. That a tub of moist coal interferes seriously with their efficiency.
3. That the feed must be uniform in quality.

These three objections are serious drawbacks to the adoption of spiral separators on an extensive scale, but the advantages they offer of cheapness and high efficiency under correct control suggest that, for special purposes, they are worthy of consideration as an alternative to other more costly processes of coal-cleaning. This is especially the case when the coal as mined is a dry coal, and where only one seam is being worked.

The objection that the spiral surfaces tend to rust on standing, and that the frictional forces brought into play are then unduly great, has frequently been over-emphasised. The trouble only arises for a short interval of time, about half an hour, after a week-end stoppage and, if other things be equal, is not proportionately greater than similar difficulties in other processes. Spiral separators have, indeed, a certain advantage that, since the process of separation and the products are visible, the adjustments necessary on starting up after a stoppage can quickly be made by the attendant.

The construction of spiral separators is illustrated in Fig. 161. In operation, the coal is fed from a shaking feeder into inclined feed shoots. In sliding down the shoots, the particles acquire a certain

initial velocity before entering the spiral column. In sliding down the spiral column, the dirt particles, because of their high coefficient of friction, do not accelerate but slide down near to the axis and are delivered at the bottom. The coal particles, however, acquire an increasing velocity and, as a result, acquire sufficient centrifugal momentum to reach the edge of the spiral surface and pass over into a separate collector.

The spiral consists of a central pillar with a series of helical plates built around it. The plates, or threads, are inclined downwards towards the central pillar, the amount of inclination being varied for different coals. The pitch of the threads may also be varied to allow for the difference. Thus, for anthracite, a pitch of 28 in. is usually employed, whereas, for bituminous coal, the spiral threads make one complete turn around the central post in 36 in. The downward inclination of the surface of the threads towards the post and the pitch are, of course, only variable in the original design, and not during operation. The slopes, once the plant is erected, are invariable.

The spiral threads, which constitute the separating surfaces, are made of steel plates with a slightly ridged or corrugated surface. Three parallel threads are arranged around the central post, and the feed shoots at the top of the separators are arranged so as to divide the material fed to them into three equal parts. On each separator, therefore, there are three separating surfaces. These may be seen in Fig. 161, which is a photograph of a battery of spirals in course of erection. In this photograph there appear to be four threads, but the lowest one is the clean coal collector. When the spiral is complete, a jacket is fixed to the lowest thread and surrounds the three upper ones. The coal passing over the edge of one of the three threads is caught by the jacket and continues to slide downwards inside it.

When the particle is sliding downwards along a thread its motion is governed by a number of forces. Imagine the particle to be situated, at any instant, on a thread midway between its rim and the axis. The forces acting on it are the forces of gravity (acting partly along the direction of the thread, and partly towards the axis), the force of friction, and the centrifugal force due to the velocity of the particle. It can be shown mathematically that, on a thread of constant inclination, if a particle begins to move outwards towards the rim (*i.e.*, if the centrifugal force exceeds the component of gravity towards the axis) it will continue to move outwards until it reaches the rim, and will not take up any path at a constant distance from the axis. To prevent shale from reaching the clean coal, therefore, the conditions must be such that the shale particles never acquire sufficient centrifugal force to cause them to mount the thread, but must be made to " hug " the axis.

There are, however, middlings particles, some of which it is desirable to include with the clean coal, and others of which must

be discarded. Middlings are dealt with by permitting them to move outwards towards the rim and retarding their progress by making them pass over a surface with a higher coefficient of friction. Their velocity, and hence their centrifugal force, is thereby reduced, and by careful adjustment some of them may be made to move back towards the axis, and others to proceed towards the rim and into the clean coal collector.

The plates with an increased coefficient of friction are arranged at intervals along the threads and near to the rim. They consist of steel plates with perforations, the perforations increasing the resistance to particles sliding over them. They are visible at A, Fig. 161. As the material travels down the thread, the middlings particles reach a position near to the rim and encounter the increased frictional resistance of the perforated plate. They are then checked and take up a course nearer to the axis. The perforated plates, or half-covers, are adjustable, so that they can be removed either partially or entirely, leaving the plain surface underneath. The amount of perforated plate exposed is adjusted so that the light middlings are not sufficiently checked to prevent them passing over the rim at some lower position in the separator, whereas the heavier middlings particles do not pass over into the coal.

To facilitate the discharge of coal particles, there are occasional V-shaped gaps in the plates, the apex of the V being nearest the axis. The gaps are provided by adjustable wing-plates, so that the width can be varied. Particles which are near to the rim are thus enabled to fall through the gaps, whereas those nearer to the axis, and therefore to the path of the true refuse, are unable to do so, but jump across to the next portion of the spiral.

Since the separation effected on spiral separators is said to be dependent solely on frictional coefficients, it would appear that these two devices for sorting the middlings are particularly arranged to deal with particles whose *surfaces* differ from those of coal and shale. If the separation is independent of specific gravity, there should be no doubt about the path of intergrown particles. If they slide on a surface of shale they should pass to the refuse ; it of coal, they should travel over the rim into the clean coal collector. The half-covers of perforated plate and wing-plates would, therefore, seem to be designed to deal particularly with such materials as the " bone " coal associated with American coals, but rarely found in English coals, and they may be of no benefit as a means of removing from the coal any particles of a high ash content which slide on a coal surface. On the other hand, with intelligent adjustment, they may be capable of correcting slight troubles arising from the different hygroscopic states of the surfaces due to changes in the atmospheric humidity, or other causes.

Although, as has been stated, adjustments are provided so that middlings particles are encouraged to pass into one or the other product, provision is made for the collection of a separate middlings

fraction. This is accomplished by means of either a baffle or a wing plate adjustment at the bottom of the separating threads, which deflects those particles which reach the bottom near to the rim of the surface into a separate collector.

In the operation of spiral separators there are a number of matters that require close attention, some of which are matters of design, others of operation. For example, the coal must be carefully and efficiently sized, for inaccuracies of screening are reflected in inefficient cleaning. Suitable sizing for cleaning British bituminous coals divides the raw coal into four grades—4 to $2\frac{1}{2}$ in., $2\frac{1}{2}$ to $1\frac{1}{2}$ in., $1\frac{1}{2}$ to $\frac{3}{4}$ in., $\frac{3}{4}$ to $\frac{3}{8}$ in. Coal below $\frac{3}{8}$ in. cannot be cleaned on spiral separators, and coal greater in size than 4 in. may be more cheaply cleaned by hand-picking.

It is also essential that spirals should never be overloaded. Each particle of material treated follows its own course down the separating surfaces, and if the feed is too rapid, coal particles may be overcrowded by dirt particles and be lost, or, on the other hand, dirt may be pushed over the rim by coal particles. The capacity of spirals depends upon the size of feed treated and the amount of coal in the refuse. With raw coal below 1 in., each separator can usually deal with 6 tons of raw bituminous coal per hour. With coal above 1 in., the capacity is about 8 tons per hour, and with the largest sizes, 12 tons per hour may be handled. The capacity is lowered, however, if the coal contains more than about 15 per cent. of refuse. These capacities, being maxima, are seldom reached in practice, because of the varying percentages of different sizes in the coal delivered to the cleaning plant.

The arrangements for feeding the coal into the spiral shoots are designed to give as regular a rate of supply as possible. The coal must be passed into the spiral shoots with only a low initial velocity. The inclination of the feed shoots is adjustable by means of a lever in such a way that, when the particles reach the spirals proper, they will be travelling with the optimum velocity. For bituminous coal, the standard equipment allows of a variation in slope from 6 to 4 in. per foot; for anthracite, the slope may be varied from $4\frac{1}{2}$ to 3 in. per foot. Because satisfactory sizing is a necessary preliminary to cleaning on spirals, fracture after screening must be minimised, and for this reason the screens should be as near to the separators as possible. A certain amount of fracture is inevitable. Particles of coal of smaller size than those for which the spiral is designed, pass down the separator in the path of the dirt. These particles are removed from the refuse by perforating the refuse-discharge shoot at the foot of the separator so that the small coal particles pass through the bottom into the shoot conveying the mixed product. This product is generally used for boiler firing, or is crushed and re-treated. In these circumstances, the small coal discharged with the refuse, either through fracture or mischance, is not wasted.

The clean coal which passes over the rim of the separating threads during passage down the spirals is collected by the outer jacket, down which it slides to a shoot at the foot. It passes thence along a belt conveyor for elevation to the loading bunkers. The middlings collected from the rim of the threads at the bottom of the spiral are passed into a second shoot for conveyance by a travelling belt to a crushing plant, and the refuse is also discharged by a shoot to a third travelling belt for removal from the building.

Spirals are arranged in batteries. A battery usually consists of eight separators, two being provided for each size of coal. The clean coal, middlings and refuse belts travel past the spirals, so that the products of each spiral in the battery are collected together. When the clean coal is to be sold on a size basis the clean product from each pair of spirals is collected separately.

In operation with certain coals and under proper supervision, spiral separators can yield a product containing not more than 2 per cent. of coal in the refuse or of dirt in the coal. For example, in two tests at Hazelrigg Colliery, and two other tests at Netherton Colliery, the average losses of coal in the refuse and of refuse in the coal were stated to be :—

	Coal in Refuse per cent.	Dirt in Coal per cent.
Hazelrigg	1·06	1·19
Netherton	1·58	1·27

The principal objections that have been raised against spiral separators have been stated. Whereas much of the criticism is justified by the inherent difficulties of the process, except with specially-selected coals, much of it may only be justified by lack of efficient control and operation. From the description that has been given it will be apparent that careful regulation of the operating details of the plant is required, and that, in cases of any irregularity, or lack of uniformity in the feed, intelligent adjustment is necessary. Under these conditions supervision should be in the hands of a skilled and experienced operator, instead of (as is common in coal-cleaning plants) in the hands of a labourer. If skilled supervision is required its cost can only be borne if the plant is a relatively large one. Given a suitable coal of suitable size and proper supervision, there is reason to believe that spiral separators are a useful and cheap alternative to wet-washing processes. In this country certain Durham and Northumberland coals and certain gas coals from other districts have been found suitable for treatment on spirals.

There are several features of spiral separation which the practical operator will view with favour. The cleaning process itself requires

no power, a feature almost unique. The only power required is for the elevation and screening of the coal and the conveyance of the products to suitable points, but these power requirements are common to most cleaning plants. Secondly, an irregular feed (provided the rate never exceeds the maximum) can be used, and the whole plant can be stopped in a moment by shutting off the feed.

In comparison with other dry-cleaning processes, the principal objections to it are its limited applicability, its inability to deal with coal below $\frac{3}{8}$ in. and the constant and careful supervision required. Moreover, with coals of a friable nature there is a considerable amount of fracture.

The Berrisford Process.—A process for the dry cleaning of coal has been worked out recently by Messrs. W. H. and S. R. Berrisford,

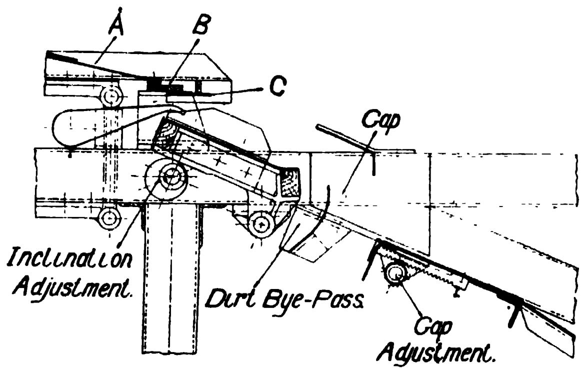

FIG. 162.—The Berrisford Process: Diagram.

and plants have been erected at Norton and Whitfield collieries, Staffordshire. At Norton, coal from $1\frac{1}{4}$ to $\frac{3}{8}$ in., and at Whitfield 2 in. nuts are treated (*Trans.* Inst. Min. Eng., 1926, 72, 97).

The method of separation is shown diagrammatically in Fig. 162. The raw coal is supplied from a jigging feed to a step, whence it falls on to an inclined plane. Coal particles have a tendency to bounce along the plane and to gather speed. Dirt particles, on the other hand, are said to have less resiliency, and slide slowly down the plane. When they reach a gap in the plane, the shale particles are moving slowly, and fall through the gap; whereas the coal particles are moving rapidly and jump across it.

A number of difficulties arose during the initial experiments, but these were overcome in the design shown in Fig. 163.

It was found to be necessary to ensure that the particles had a uniform initial velocity. This has been achieved by feeding the raw coal from a jigging pan A (Fig. 162) on to a stationary step B, a row of particles being left on B when the jigger is withdrawn. On the forward stroke of the jigger the row of particles on the step

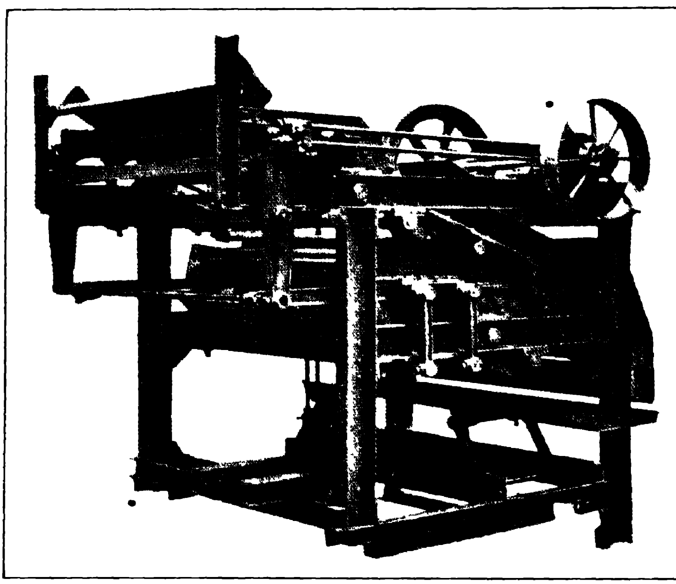

FIG. 163 The Berrisford Separator.

[*To face page* 372

is pushed on to an extension piece C, actuated by the jigging mechanism. When the jigger moves backwards a second time the member, C, is withdrawn beneath the stationary step and the particles fall on to the plane from a constant height, and with a uniform speed.

The surface of the inclined plane consists of a glass plate, and its optimum length has been determined as the result of repeated trials. With planes which were too long, the coal caught up the dirt sliding down in front of it and the collisions interfered with their independent progress. If the plane were too short, the coal was still bouncing when it reached the gap, and occasionally fell into it. The difficulty was overcome by bending the lower end of the surface in a horizontal direction to form a lip. As it reaches the lip the bouncing motion of the coal is destroyed, and both the coal and the dirt particles slide forward. The lip has the further advantage of reducing the speed of the dirt, which is always sliding already, by a greater amount than it reduces the speed of the coal.

The gap is divided into two portions by a division plate. The particles falling into the first portion of the gap are " pure " dirt ; those falling in the second part contain middlings, together with some coal and shale. These are passed to a second plane beneath the first, where they are separated at a second gap.

The inclination of the plane can be varied by an eccentric adjustment, and the width of the gap by a rack and pinion. Experimental data are given in Table 104.

TABLE 104.—OPERATING DATA. BERRISFORD SEPARATOR

Size of Coal in.	Height of Fall in.	Length of Slide in.	Length of Lip in.	Width of Gap in.
2– 1	5	12	2	8
$1\frac{3}{4}$– $\frac{3}{8}$	4	12	2	7
$\frac{1}{2}$– $\frac{1}{4}$	2	9	$1\frac{1}{2}$	4
$\frac{1}{4}$– $\frac{1}{8}$	1	$4\frac{1}{2}$	1	$3\frac{1}{2}$
$\frac{1}{8}$– $\frac{1}{32}$	$\frac{7}{8}$	$2\frac{3}{4}$	$\frac{1}{2}$	1

At Norton, with coal from $1\frac{1}{4}$ to $\frac{3}{8}$ in., the average ash content of the clean coal for four months was 7·3 per cent., the raw coal containing " 30 per cent. of free dirt." For coal above 1 in. in size, a machine occupying the space of a 7 ft. cube has a capacity of 15 tons per hour. It is necessary to size the coal between fairly close limits before cleaning.

Provision is made for an air-blast over the surface of the inclined plane. The air current created is not an essential part of the method of separation, but is provided to dislodge any dust which

collects on the glass surface, and which might interfere with the efficiency of the process. No provision appears, as yet, to have been made to collect this dust; its amount is said to be slight, and the amount of fracture of the coal is also said to be almost negligible. At present the process is not used for coal below $\frac{3}{8}$ in. in size, and its principal drawback in operation is a high loss of coal in the refuse. The Berrisford plant shares with spiral separators the advantages of requiring no power and being able to work with an irregular or non-uniform feed.

The Etna separator recently installed at one of the Baddesley Collieries Ltd.'s pits is similar in principle and general design to the Berrisford separator.

The Dry Coal Cleaning Co.'s Process.—In this process, introduced recently in England, the separation of dirt from coal takes place in two stages, in which different principles are employed.

The primary stage is accomplished in wooden troughs, shown in Fig. 164, which is a photograph taken whilst the machine was in operation in a South Wales colliery. The raw coal is fed from the shoots at the back of the building shown in Fig. 164 into the inclined troughs, in which it spreads out into a thin layer, practically all the particles coming in contact with the surface. The trough is given a jerking movement, backwards and forwards, by means of a crank shaft, and the particles move down the plane under its influence. In so doing they are thrown against a series of converging wooden baffles, spaced at intervals in their path, which cause the edges of the layer to converge and form a deeper bed of material. As the particles squeeze together, those of the lighter material (coal) mount upon the heavier ones and the bed thus becomes stratified. The light particles in the upper layers of the bed pass over a horizontal diaphragm and travel further along the trough, where the process of spreading, convergence and mounting is repeated. The heavier particles in the lower layers, however, pass under the diaphragm and are led into shoots to be recleaned in the secondary stage.

These shoots, and the apparatus for the secondary stage of the same plant, are shown in Fig. 165. There are four shoots, each leading to a separate table. Each table is given a rapid transverse oscillating motion, across the line of its inclination, and the particles first of all spread out under its influence and then travel diagonally across the table against a barrier. The resistance of the barrier is said to cause a rebound which spreads out the coal according to density differences. The angle of inclination is adjustable.

In both stages of the process, a current of air is blown on to the particles, assisting them in their movement and tending to keep the bed in a loose condition. The quantity of air used is about 4,000 cub. ft. per hour.

Coal below $1\frac{1}{2}$ in. is treated without preliminary classification. In the plant in operation, a unit consists of two primary troughs and

FIG. 164 — The Dry Coal Cleaning Company's Process Primary Stage

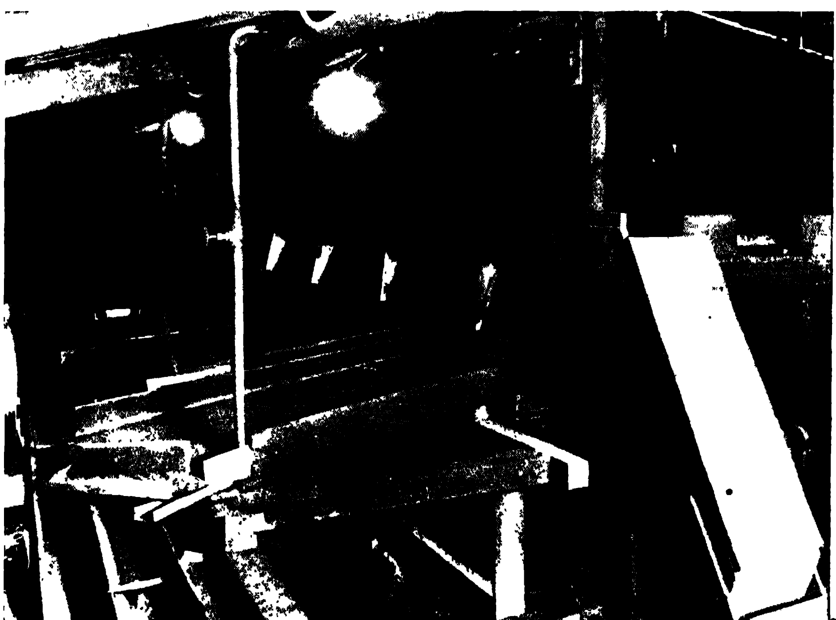

FIG 165 - The Dry Coal Cleaning Company's Process Secondary Stage.

To face page 374.

eight secondary tables, but, for most coals, ten secondary tables are recommended for each primary trough. The capacity of such a unit would be about 10 tons per hour on coal below 1 in. A power consumption of about 2 h.p. is required to actuate the apparatus, a stroke of about 1 in. being imparted 350 times per minute.

The plant described has been modified considerably since erection and the primary troughs now effect no classification. In future plants, only the larger sizes of coal, over 1 in., will be recovered by the primary process, all the material below 1 in. passing to the secondary tables for separation. The finest dust is removed from the refuse, and this may be mixed with the coal, or divided by screening into further fractions, some of which can be mixed with the clean coal.

CENTRIFUGAL SEPARATION

The principles of centrifugal separation have been employed to remove dirt from coal, but without marked success. The feed, which must be sized, is fed on to a rapidly-revolving disc and is projected radially from it. The trajectories of the particles depend upon the momentum which they have acquired when they leave the disc, and the large and heavy particles are thrown a greater distance than the small and light ones. The products can thus be collected in a series of concentric troughs. This was the method employed in the Clarkson-Stansfield separator used in Wales in 1890. In the Pape-Henneberg separator, the separation was assisted by a gentle centripetal air current.

Several types of enclosed centrifugal separators have been tried for various purposes, but, except for the removal of water or colloidal material from slimes, they have been found unsatisfactory, owing largely to the difficulty of making the machine continuous in its action.

CHAPTER XX

PROCESSES USING MEDIA OF HIGH DENSITY: THE CHANCE PROCESS

PROCESSES for coal cleaning which make use of a separating medium having a higher density than water are frequently called flotation processes. The use of this term is apt to be confusing, for "flotation" is established as an abbreviation of the term "froth flotation." Though this abbreviation may be unfortunate, there are certain grounds to justify its use. The aggregation of air bubbles and solid particles which collects at the top of a froth-flotation cell has a specific gravity less than that of water, and it rises to or floats on the surface because of its lower specific gravity. Once the association of the particles with the air bubbles is accomplished, the process is strictly one of flotation.

In the Chance process, which at present is the only commercial process employing a medium of higher density than water, the separation of coal from dirt is effected in an upward current of a medium of sand and water, and it is partly a flotation process and partly a process of upward-current classification. It would therefore seem advantageous to discard the use of the term flotation as applied to the Chance process, and to confine its use to those of froth flotation, and to any future processes in which separation of raw coal into a floating portion and a sinking portion is effected in a body of fluid which, to all intents and purposes, is still.

HISTORICAL

The idea of separating coal from dirt by using a liquid of high specific gravity, in other words, of doing float and sink experiments on a large scale, was first suggested by Sir Henry Bessemer in 1858, though Bérard had used it on a small scale for examining coals before washing. Since then the method has appealed to a number of inventors, but, despite the simplicity of the principle and the high efficiency to be expected, the results obtained hitherto (the Chance process excepted) have been disappointing on account of difficulties of operation.

Sir Henry Bessemer (B.P. 1724, 1858) proposed to remove the impurities from coal by "immersion in a tank or bath containing a fluid, the specific gravity of which is greater than pure coal and less than the substances to be separated therefrom." In his process, the floating particles were removed by revolving or reciprocating skimmers or rakes and the sinking particles by buckets or by an

endless screw or revolving sluice, the bottom of the vessel being cone-shaped to aid the collection of the sinking particles at their discharge point. By these means the process was made continuous. The fluid adhering to the separated particles was removed by filtration, or drainage, or in centrifugal dryers.

The fluid used consisted of any cheap liquid and a solution of iron dissolved in muriatic acid (ferric chloride) was found to be suitable. Manganese chloride and barium chloride were also cited, but the best solution was calcium chloride. When metallic salts were used, the solution recovered by washing the clean coal and refuse with water was concentrated by evaporation for further use. The clean coal was to be used in the raw state with or without the addition of pitch or of lime or for coking purposes.

A similar process was used in Germany in 1859 by Englinger, and by Brown in America in 1884. A large plant was also erected at the Laura and Bolhörst mine near Minden, Germany, but after a long trial it was abandoned.

In all these methods an aqueous solution of calcium chloride or other suitable salt was employed. In 1918, however, the Chance process was devised, in which the medium of high specific gravity was obtained by maintaining a suspension of sand in water, so that the mixture behaved as a fluid body with a density higher than that of water. The sand was easily removed from the products and returned to the separating vessel, and continuity was assured by causing the clean coal to enter at one side of the vessel and to overflow at the other side in a current of the separating medium.

THEORETICAL CONSIDERATIONS

It is unnecessary to consider the theory of separation of coal from dirt in a solution of calcium chloride, the separation then depending solely upon differences of specific gravity. In the Chance process, however, the required density is obtained by maintaining sand grains in suspension in an upward water current, and the influence of the size of the coal and dirt particles must then enter into consideration.

It has been shown in Chapter III that the speed of an upward current of water required to prevent a given particle from falling against it may be expressed in the form

$$V = K\sqrt{r(s - 1)},$$

where V is the current speed, K is a constant, r is a linear dimension of the particle, and s its specific gravity, the specific gravity of water being unity. From this formula it may be calculated that a water current of speed 0·82 ft. per second will support coal particles of S.G. 1·3 up to 1 in. in size, but will not support shale of S.G. 2·6, and of size greater than 0·2 in. In a liquid of S.G. 1·25, a current

of the same speed will support coal particles up to a size of 7·5 in. and shale to a size of 0·3 in.*

It will be noted that, with an increase in the specific gravity of the medium from 1 to 1·25, the size of shale particles supported is increased slightly from 0·2 to 0·3 in., but the size of coal is increased considerably from 1 to 7·5 in. In a current of a fluid of S.G. 1·25 moving upwards with a speed of 0·29 ft. per second, all coal of size 0 to 1 in. could be separated from shale of size 0·045 to 1 in. This shows that by using a fluid of higher specific gravity than water, a much lower current speed can be used (0·29 ft. per second instead of 0·82 ft. per second) and a wider range of sizes can be separated by it (0·045 to 1 in. instead of 0·2 to 1 in.).

Moreover, if the fluid had a density more nearly approaching that of coal, say, 1·28, the separation is further facilitated, for a current speed of 0·23 ft. per second would separate all the shale from coal of size 0·027 ($\frac{1}{40}$) to 1 in.

In these calculations it is assumed that the value of the constant K remains the same when water is replaced by a mixture of sand and water, and also that the motion of the particles is unrestricted by the presence of other particles. Actually the particles of dirt sink through and into a bed of other dirt particles, and in these circumstances it is not unlikely that the ratio of separable sizes is increased.

The current speeds and sizes given are arbitrary ; they are not absolute, but they are relative, and they illustrate the principles upon which the Chance process depends. They show, for example, that a process such as the Chance can deal with an unsized feed, whereas in a process such as the Draper, in which an upward current of water (S.G. 1) is used, it is necessary to size the coal before washing between limits of size in the ratio of about 3 to 1. They show also that processes using media of specific gravity greater than that of water are capable of control in two respects, firstly, the effective density of the fluid employed, and, secondly, the speed of its upward movement.

THE CHANCE PROCESS

The Chance process for washing coal was designed originally for cleaning American anthracite, but, in October, 1925, a plant was erected at Mount Union, Pennsylvania, for treating bituminous coal. For the latter purpose, the operation of the process differs in the specific gravity chosen for the medium. The general lay-out of the plant is similar, but integral parts of it are differently dimensioned.

In America the bulk of the anthracite produced requires preparation before it can find a satisfactory market. A hundred years ago, only the thick and clean seams of anthracite were mined and material

* The values of K given in Table 43, Chapter VI, are used for these calculations.

below $\frac{3}{4}$ in. was discarded. Since that time the smaller sizes have been used until, at the present time, practically the whole output is marketable, though the portion below $\frac{1}{16}$ in. has no ready market. As in this country, anthracite must be carefully sized before shipping, and the standard sizes to-day are :—

	In.
Egg	$3\frac{7}{16}$–$2\frac{1}{2}$
Stove } Nut	$2\frac{1}{2}$–$1\frac{9}{16}$
Chestnut } Nut	$1\frac{9}{16}$–$\frac{11}{16}$
Pea	$\frac{11}{16}$–$\frac{1}{2}$
Buckwheat No. 1	$\frac{1}{2}$–$\frac{1}{4}$
Buckwheat No. 2 (Rice)	$\frac{1}{4}$–$\frac{3}{16}$
Buckwheat No. 3 (Barley)	$\frac{3}{16}$–$\frac{1}{16}$
Buckwheat No. 4	$\frac{1}{16}$–0

In general, American anthracite may be classed as a high-ash fuel. In many seams the smaller sizes of the run-of-mine coal contain more ash than the larger sizes ; in others, on the other hand, the larger sizes are the highest in ash. Frequently the ash content decreases with size to about 28 mesh, below which the ash content increases. The standard method of preparation is to hand-pick the sizes larger than nut, to crush the rejected material and remove the dirt by mechanical means or by washing. It is generally necessary to clean mechanically all the anthracite below nut size if it is to be readily saleable.

For many years, jig-washers treating closely-sized material, were used for washing. They were troublesome and often inefficient when treating the smallest sizes, and the introduction of the Chance process in 1921 enabled better preparation to be effected than had usually been possible in jig-washers. It had the advantages of higher efficiency and the ability to deal with unsized material. That it was unable to clean material below $\frac{3}{16}$ in. was of little importance, because jig-washers were inefficient for these sizes, and because there was little demand for them. Moreover, the introduction of the Deister-Overstrom table about the same time enabled the difficulty to be overcome.

In the Chance process the raw coal is fed into a conical separating chamber (Fig. 166), which contains a mixture of sand and water, the sand being kept in suspension in the water by an upward water current and by a revolving agitator. The light material, which it is required to recover, floats near to the surface of the fluid mixture and travels round the cone until it reaches the discharge point. The dirt sinks through the separating medium and falls through a classifying column into a refuse chamber, from which it is periodically discharged.

By varying the relative proportions of sand and water in the separating cone, any desired specific gravity of the mixture can be obtained between 1·25 and 1·8. When the two are present in the

correct proportions to give the effective specific gravity required, no difficulty is found in maintaining the same conditions, sand and water being constantly added to replace that which overflows with the washed coal.

It has already been stated that the mixture is agitated by a current of water, which is admitted at the base of the cone. Because of the conical shape of the vessel the speed of the upward water current varies at different levels in the cone, being faster near the base. The sand, because of its high specific gravity, tends to

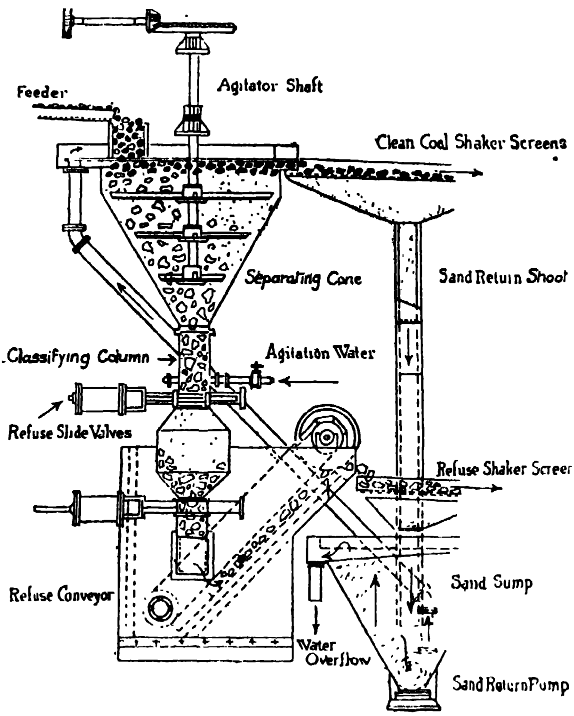

FIG. 166.—The Chance Process: Diagram.

settle, and if the speed of the water current near to the top of the cone is too low the sand falls out of suspension from the upper layers of the mixture to a lower region, where the current speed is great enough to support it. For any given current speed, the rate of loss of sand with the washed coal is constant (sand being added to make up the deficiency). If the current speed falls, less sand is discharged, and, the supply of fresh sand being maintained, the mean effective specific gravity is increased. If the speed of the current is increased, an excessive loss of sand occurs and the mean specific gravity falls.

In practice, the specific gravity is controlled by regulation of the amount of water added. The amount added, and consequently the

speed of the upward current, is just sufficient to retain the proper amount of sand and to maintain the required specific gravity. The specific gravity of the upper layers is determined directly during operation by the use of a hydrometer or of specific gravity balls. These balls are made of different densities and the specific gravity of the medium is determined by seeing which of them sink and which float. For washing anthracite, specific gravities between 1·6 and 1·8 are used, according to the nature of the coal, and with bituminous coal the density ranges from 1·3 to 1·5.

The coal is fed into the cone from a shoot in a stream of water. The lip of the shoot is placed only a short distance above the water surface so that the particles slide into the cone rather than fall into it, and the direction of the feed is tangential to the rim of the cone, so that the water, which is revolved by a stirrer, is not unduly disturbed. As shown in Fig. 166, the coal may be allowed to enter the washer through a short vertical column, which is kept filled with coal to remove the possibility that particles may fall directly into the separating cone with an appreciable velocity. The feed can consist of unsized material below 3 or 4 in.; the fine dust must, however, be removed before washing. It is frequently advantageous to divide the feed into two sizes, say, above and below 1 in., the two fractions being fed into the cone by shoots at different points. If the feed consists of an assortment of all sizes up to 4 in., all fed in one stream, some of the small refuse particles may be mechanically entangled amongst the large coal particles and be driven round the cone with them to the clean-coal discharge point. It follows, then, that separation will be facilitated if a rough grading of the particles is made and the two fractions be fed separately. A large refuse particle can more easily sink through a layer of small coal than can a small refuse particle through a layer of large coal. The fraction comprising the small sizes should therefore be fed into the cone before the fraction containing the larger sizes.

It is not essential to grade the coal in this manner before it is fed into the washing cone; it is merely practised for mechanical convenience and to increase the capacity, especially with a coal high in material of about the same specific gravity as the medium, or when the washer is overloaded. The sizing need not be particularly accurate. The washing process does not require preliminary sizing in the same manner as do many other processes (*e.g.*, the Draper); in the Chance process the sizing may be considered as an additional capital cost to increase the earning power of the capital already expended in the plant.

In many plants more than one cone is employed, according to the output required. Where this is the case, one cone is fed with the material above about 1 in., and the other with the smaller sizes.

The feed shoots are wide, so that the raw coal is fed in a shallow stream of water and spreads out into a thin layer on or just below the surface of the washing medium. This allows the upward

current of the fluid to flow around each particle, so that each behaves as a unit, sinking if its specific gravity is higher than that of the medium, and floating if it is lower. No difficulty is then experienced through aggregates of coal and dirt floating together across the surface.

The coal usually floats round the cone a few inches below the surface of the fluid medium in the separating cone. The supernatant fluid consists of fairly pure water, associated with only small quantities of sand, the bulk of the sand being concentrated in lower layers. When the coal reaches the discharge point, it passes over a lip in the rim of the cone and falls on to a desanding shaker screen. The overlying water, which is almost free from sand, then flows through the layer of coal and acts largely as washing water, removing the bulk of the sand with it through the apertures in the screen. Further along the screen the clean coal is sprayed with fresh water to remove the remainder of the sand. After the sand has been removed, the coal passes over sizing screens and is discharged into storage bunkers or directly into wagons.

The dirt, which sinks through the separating medium, falls to the base of the cone and passes into the classifying column. The water used for agitation enters at the lower end of this column, and since the column has a relatively small cross-sectional area, the water current travelling up it has a relatively high velocity. The rapid water current not only removes any occasional pieces of coal which have fallen with the dirt, but it also removes most of the sand from the refuse. The refuse is allowed to collect at the bottom of the classifying column and, periodically, a sliding valve is opened to allow the accumulation to fall into a refuse-collecting chamber. A second sliding valve, which is only opened when the upper valve is closed, allows the refuse to fall periodically from the collecting chamber into the boot of the refuse elevator, which conveys it to a desanding screen. The sand associated with the refuse is washed through the screen by a water spray.

The sand washed from the clean coal and from the refuse falls down a vertical shoot into a sand sump. In the sump there is only a gentle upward current of water, and the sand settles to the bottom, whence it is pumped to the top of the cone and returned to circulation. The water introduced into the sump is that used for the desanding operation. A portion of it is pumped by the sand pump and returned to the cone with the make-up sand, and the remainder overflows from the sump. The water sprays on the clean coal and on the refuse desanding screens not only remove the sand grains but also those small particles of coal and dirt produced by fracture of larger particles during the washing operation. The fine coal and dirt produced either pass out of the system in the water overflowing from the sand sump or are returned to the cone with the sand. The bulk of the fine particles overflows from the sand sump; the coal because it is light enough to be carried upwards in the water current,

and the dirt because it usually arises from the disintegration of shale and (being almost microscopic in size) remains in the water as a suspensoid. The overflowing water is either clarified and re-used, or is run to waste.

The sand returned to the cone is introduced in a slow stream and is distributed evenly across a radius of the cone. It is very desirable that the surface of the fluid separating medium be as little disturbed as possible, as otherwise the coal may sink to a depth which prevents its ready discharge over the lip. It is also desirable that the sand be introduced at a uniform rate, for the inflowing sand and water help to carry the floating coal round the cone.

The surface of the fluid becomes disturbed if there is an air leak at either of the refuse valves. This is especially objectionable if the leakage occurs at the lower valve, for, in these circumstances, air collects in the refuse-collecting chamber and rises suddenly through the fluid mass immediately the upper valve is opened, causing a disturbance throughout the cone. Vertical water currents are then produced, which tend to carry coal downwards with the dirt and also cause an excessive discharge of sand with the clean coal, thereby lowering the specific gravity of the fluid. If the leak occurs at the upper valve, the bubbling produced is only slightly less objectionable, but it can be more easily noticed and rectified. To avoid air-leaks it is important to maintain a sufficient head of water in the chamber at the foot of the refuse elevator, so that no air can pass into the refuse-collecting chamber when the lower discharge valve is opened. In the latest plants the possibility of these air-leaks is eliminated by surrounding the valves with a hydraulic chamber containing water under pressure and the refuse valves are operated hydraulically.

The cone itself is constructed of mild-steel plates riveted together. Its dimensions depend upon the nature of the coal and the capacity required. The actual process of washing takes place in the upper portion, where the vessel is cylindrical and not conical in shape. The lower conical portion is not a necessary part so far as the actual washing is concerned, but it serves several purposes; firstly, it causes the refuse to fall to a central point, whence it is readily removed without carrying large quantities of sand with it, and secondly, it provides a long path for the upward water current, so that the water is evenly diffused throughout the mass and does not disturb the quiet surface. The conical shape also results in a variation of the speed of the upward current, and therefore a variation of the concentration of sand in different horizontal zones of the fluid. In the uppermost layers the current is too slow, when the washer is working properly, to support a large quantity of sand; in the lowest layers it is so rapid that little sand can fall downwards against it. The highest sand concentration, therefore, occurs at some intermediate zone. Consequently, the clean coal only carries

away small quantities of sand from the washer, and the falling refuse passes through a zone in which much of the sand is washed upwards from it.

The revolving agitator has three projecting arms and is driven, relatively slowly, through bevel gearing. Its function is twofold. It causes a rotary motion of the fluid mass which assists in mixing it and in carrying the floating coal round the cone to its overflow, and secondly, the revolving arms scrape away any sand or refuse that accumulates on the inclined inner surface of the cone and breaks up any sandbanks.

Ordinary river or seashore sand, or crushed sandstone are suitable for use in the Chance washer. These materials may vary slightly in density, and allowance must be made for possible variation when the plant is designed. The usual specific gravity of these solids is 2·6. The sand most suitable for use is that which passes a 40-mesh sieve, but remains on a 60 mesh (Tyler standard). There should be only a small proportion finer than 80 mesh or coarser than 30 mesh.

Fine sand is difficult to settle for re-use, but it is also unsatisfactory because it tends to accumulate in the upper layers of fluid in the washing cone and to be washed over the coal-discharge lip. The accumulation of fine particles prevents the specific gravity lower down the cone from attaining its proper value, and the clean coal particles therefore descend further into the fluid medium than they should and are not discharged at the clean-coal overflow as fast as fresh coal is supplied. The coal then accumulates in the separating cone, causing an excessive overflow of sand.

If the sand is too coarse, more agitation is required to keep it in suspension and the large particles sink to the bottom with the refuse. Excessive discharge of sand results, with greater wear of the valves, pumps and conveyors.

It is most important that excessive losses of sand be avoided, otherwise, make-up sand being supplied at a uniform rate, there will be a deficiency of sand in the cone and the mean specific gravity of the fluid will fall, with a consequent loss of coal in the refuse. An excessive discharge of sand from the top of the cone results from the use of too great a proportion of fine sand, or a disturbance of the fluid medium and the creation of eddy currents. Sand is discharged from the base of the cone in excessive quantities if the sand is too coarse, or if the water current is irregular. An undue amount of sand may also be lost if the rate of feed of raw coal is very irregular or differs widely in quality. This has been overcome in the latest plants by using a large sand-water storage sump and by circulating the mixture in a closed system, controlled automatically.

The capacity of a Chance washer depends essentially upon the quality and size of the raw coal. The cones are made in two standard sizes for anthracite washing, the larger having a maximum diameter (at the top) of 15 ft., the smaller cones having a diameter of 7½ ft.

Occasionally, cones with a diameter of $13\frac{1}{2}$ ft. have been installed. In bituminous coal-washing practice, cones 10 ft. in diameter are used. In cleaning anthracite, or a coal of which only 20 to 25 per cent. is smaller than 1 in. and which contains 10 to 15 per cent. of refuse, the capacity may reach 200 tons per hour in the 15 ft. cone. The smaller cone can deal with about 70 tons per hour. The capacity with a clean coal is limited by the rate at which washed coal can be discharged ; with a dirty coal the limiting factor is the rate of discharge of refuse through the refuse valves.

The first Chance plant was erected for the Grand Tunnel Company, at West Nantichoke, Pa., in 1921. The plant consisted of one cone 7 ft. 6 in. in diameter, capable of cleaning about 60 tons per hour. The results show that, on the average, the clean coal contained 1·85 per cent. of dirt, and the average result of 747 tests on the refuse show that $1\frac{1}{2}$ per cent. of the refuse consists of coal floating in a liquid of S.G. 1·75 ; this is equivalent to a loss of 0·22 per cent. of the coal fed. These results, obtained in the first plant erected, are highly satisfactory.

The Rose Coal Company erected a plant in 1921 for the re-treatment of jig refuse at Winton, Pa. During one period of fifteen months' operation, 242,000 tons of refuse were treated, yielding 39,129 (16 per cent.) of marketable anthracite from material which was thrown away when the coal was washed in jigs.

Up to 1926 twenty-four Chance plants had been erected with a total capacity of 4,225 tons per hour, or an average hourly capacity of 176 tons. The largest plant was erected for Madeira Hill & Co., its capacity being 1,000,000 tons per year. The plant includes four separating cones, each 13 ft. 6 in. in diameter (type F), and the coal from $4\frac{7}{16}$ to $\frac{1}{16}$ in. is treated. The raw coal is a mixture of ten separate seams, varying in specific gravity, ash content and mode of fracture, and it is a considerable recommendation for the Chance process that, with the varied feed that must of necessity arise from time to time when such a number of seams are worked, the plant is able, on the average, to produce domestic coal (above $\frac{3}{4}$ in.) containing frequently less than one-half of 1 per cent. of refuse and seldom as much as $1\frac{1}{2}$ per cent. The average loss of coal is 2 per cent. of the refuse, or less than half of 1 per cent. of the coal fed

The general manager of the company estimates that the cost of erection of the plant was only 70 per cent. of that of a jig-operated plant, and that the cost of operation is 10 to 20 per cent. per ton less. In addition, the plant recovers $3\frac{1}{2}$ per cent. more coal than did the former jigs (*Mining Congress Journal*, 1927, 13, 205).

The lay-out of the plant at Mount Union, Pennsylvania, for the washing of bituminous coal is shown in Fig. 167. It was put into operation in October, 1925. The plant contains two cones, each 10 ft. in diameter, in one of which the coal from $4\frac{1}{2}$ to 1 in. is washed, the 1 to $\frac{3}{8}$ in. coal being washed in the other. The coal

below $\frac{3}{8}$ in., containing 10·4 per cent. of ash, is by-passed and mixed with the washed coal. Each cone has a capacity of 150 tons of coal per hour, and the inclusive cost of washing is less than 5 cents per ton (of 2,000 lb.). The complete plant for preparing the run-of-mine coal deals with about 500 tons per hour, but only about 300 tons is passed to the washing section, the remaining 200 tons being hand-picked or by-passed.

The raw coal entering the plant passes over conveyors to a 6 ft. shaking screen which divides it into $4\frac{1}{2}$ in. lump and undersize. Provision is made to divide the lump, if desired, into over and under 10 in. The $4\frac{1}{2}$ in. undersize is passed by a scraper conveyor to four Shiley double-deck jigging screens which divide it into domestic ($4\frac{1}{2}$ to 1 in.), stoker (1 to $\frac{3}{8}$ in.) and fine ($\frac{3}{8}$ in. to 0) coal. The domestic and stoker sizes are conveyed to the washing cones, vibrating guard screens being installed immediately before the cones to remove dust caused by fracture. The $\frac{3}{8}$ in. to 0 coal passes directly to a mixing conveyor for loading without washing

The clean coal from both cones passes over combined desanding and sizing screens, making products $4\frac{1}{2}$ to 2 in., 2 to 1 in., and under 1 in. The dust caused by fracture is mixed with the under 1 in. fraction. The two larger sizes of washed coal are loaded into wagons or are mixed together, and with the under 1 in. and the unwashed $\frac{3}{8}$ in. to 0 coal on a mixing conveyor, from which it can be loaded directly into wagons or passed back to the tippler and mixed with the coal above $4\frac{1}{2}$ in.

The refuse is removed from the cones, and passes, by means of a common water-sealed conveyor, to a desanding screen, whence it is loaded into wagons. No refuse hopper is required, because sufficient refuse can be stored in the cones themselves to permit a full wagon to be replaced by an empty one.

The sand sump is a circular concrete tank, 21 ft. 6 in. in diameter. The circulating water overflows from the rim to a water sump, from which the water for circulation and for desanding is taken by a 2,500 gallon centrifugal pump. The thickened sand is returned to circulation from the sand sump through a pair of 1,000 gallon centrifugal sand pumps.

Make-up water is furnished by a centrifugal pump, and some of the water from the water sump is allowed to run to waste, carrying with it a fine slime, high in sulphur.

The total operating cost during the year 1926 was under 8 cents per ton, with the plant working at an average daily capacity of 60 per cent. of its maximum. Of the cost of 8 cents per ton, more than 3 cents is required for labour at the tipple, and for hand-picking, and would be incurred whether the coal were passed to the washery or not. Much of the remaining cost of 5 cents would still be incurred for screening and loading if the washery were omitted. The whole plant for preparing 500 tons per hour, including picking, washing and screening, cost $140,000 (under £30,000).

The whole plant requires 460 h.p. For the washery, 240 h.p. are required, this being made up as follows :—

	H.p.
Cone agitators, refuse conveyors and de-sanding screen	75
Clean-coal desanding screen . . .	30
Two 6 in. sand pumps	50
10 in. water circulating pump . . .	50
Refuse valve pump	20
Make-up sand and water	15
	240

Excluding the by-passed coal below $\frac{3}{8}$ in., the amount handled is about 260 tons per hour, so that less than 1 h.p. per ton of hourly capacity is required.

The washing plant differs in certain details from the standard type employed for anthracite treatment, because a lower density of

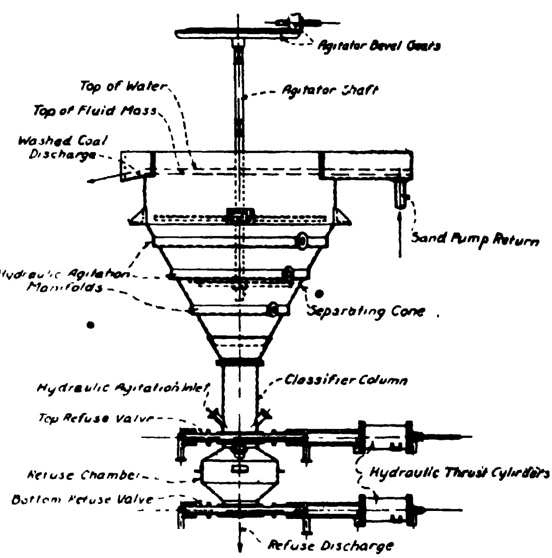

FIG. 168 Chance Washing Cone for Bituminous Coal.

the fluid medium is required. To obtain the lower density, a greater amount of water must be added, and the arrangements for its addition demand modifications to the design. The water is added through the throat or classifying column, and its diameter controls the speed of the water current passing through it. If the current is too fast, not only will the sand particles be scrubbed from the falling refuse, but the smallest refuse particles may also be retained in the cone. The throat must, therefore, be wide in order to give a fairly

slow current. On the other hand, the water current must not be too slow or an excessive amount of sand will be discharged with the refuse.

In the Mount Union Plant, with 10 ft. diameter cones, the 22 in. throat used on the 13 ft. 6 in. diameter anthracite washing cones is adopted. This enables more water to be added per sq. ft. of surface area at the top of the cone, where the diameter is constant, and where all the separation takes place. The quantity of water added through a 22 in. throat is, however, insufficient for the proper operation of the plant (the speed of the water in the throat being limited, as previously stated). Three perforated water pipes are therefore placed in the cone at different levels to supply additional water. The arrangement of the cone is shown in Fig. 168.

Because the separating medium has a greater fluidity with the low densities (1·3 to 1·5) required for washing bituminous coal than with the higher densities (1·6 to 1·8) used for anthracite washing, the revolving agitator is less rigidly constructed. It consists of three steel arms attached to a $4\frac{7}{16}$ in. shaft, actuated by bevel gearing. Less provision is also required for make-up sand and for sand return, but greater water pumping capacity is necessary. It has been found in practice that, with the relatively large amount of water to be circulated, irregularities of the feed are of practically no consequence when washing bituminous coal, and, at Mount Union, no raw coal hoppers are utilised.

The results of a test on 442 tons of coal washed at Mount Union were as follows :—

	Size (in.).	Per cent of Total	Ash per cent.	Sulphur per cent.
Feed coal	$4\frac{1}{2}$-1	62·8	11·30	1·55
	1 - $\frac{3}{8}$	37·2	10·36	2·32
Washed coal	$4\frac{1}{2}$-1	—	7·40	1·44
	1 - $\frac{3}{8}$	—	7·53	1·37
Refuse	$4\frac{1}{2}$-1	—	61·55	4·84
	1 - $\frac{3}{8}$	—	58·45	13·15

The coal below $\frac{3}{8}$ in. at Mount Union contains 10 per cent. of ash, and this is by-passed without washing. Bituminous coal of sizes down to $\frac{3}{16}$ in. can be washed, as in anthracite plants, but it was considered that to wash coal below $\frac{3}{8}$ in. was uneconomical in view of the cost of settling tanks and the resultant increase in the moisture content of the washed coal. By mixing this material in a dry state with the 1 to $\frac{3}{8}$ in. washed coal, it is possible to ship the products with 1·9 per cent. of moisture in the larger sizes ($4\frac{1}{2}$ to 1 in.) and 6 per cent. in the coal below 1 in. If the coal below $\frac{3}{8}$ in. were

washed, the moisture content would be increased by at least as great an amount as the ash content could be reduced. During 1926, tests have been conducted washing the coal above $\frac{3}{16}$ in., and the moisture content of the entire product (4 to $\frac{3}{16}$ in.) was then 4·5 per cent.

The floor space required for a Chance washery is remarkably small. The Mount Union plant covers an area of 6,404 sq. ft., of which only 1,904 sq. ft. are devoted to the actual washing and screening plant. This is equivalent to 6 sq. ft. per ton of hourly capacity. The height of the building required is 37 ft., including space for loading into railroad cars.

The chief objection to the Chance process is that it cannot be used for coal smaller than about $\frac{3}{16}$ in. This limit applies not so much to the coal as to the dirt. The speed of the current of water admitted at the base of the cone must be such that it maintains the sand in suspension in the body of the cone. It will also, therefore, maintain particles of dirt in suspension, if they are of the same size as the sand grains and have the same specific gravity. At the throat of the cone, where the current speed is greatest, it is impossible for particles of dirt to fall against the current unless they are considerably larger than the grains of sand in the body of the cone. Consequently all those particles of dirt must be removed from the raw coal fed to the washer, which cannot be separated from the sand suspended in the cone. There is scarcely any limit to the size of coal that can be fed, but there is a definite limit to the size of dirt.

Suppose that it is desired to remove all the material of S.G. 2 or more from the raw coal and that the sand has a S.G. of 2·6. The velocity of a current of water which will maintain in suspension a sand grain of size r_1 may be written

$$V_1 = K\sqrt{r_1(2{\cdot}6 - 1)} = K\sqrt{1{\cdot}6\,r_1}.$$

Suppose that, at the base of the cone, the speed of the current is twice that at a higher level of the cone at which the bulk of the sand is suspended. Then, for a dirt particle of S.G. 2 and size r_2,

$$2V_1 = K\sqrt{r_2(2{\cdot}0 - 1)} = K\sqrt{r_2}.$$

From these equations,

$$\frac{r_1}{r_2} = \frac{1}{4 \times 1{\cdot}6} = \frac{1}{6{\cdot}4}.$$

In practice, the largest sand grains have a diameter of about $\frac{1}{50}$ in. Putting $r_1 = 0{\cdot}02$, from this equation, $r_2 = 0{\cdot}128$, or about $\frac{1}{8}$ in. This calculation, were all its assumptions correct, would give a lower limit of size of $\frac{1}{8}$ in. In washing anthracite, when the dirt to be removed nearly always has a S.G. of 2 or more, a limit of about $\frac{3}{16}$ in. is required.

In washing bituminous coal, the lightest dirt has a specific gravity of about 1·6, and in these circumstances, putting $r_1 = 0{\cdot}02$, the

value of r_2 becomes 0·213, or about $\frac{7}{32}$ in. This suggests that the smallest size of bituminous coal that could be treated would be in the neighbourhood of $\frac{1}{4}$ in., but this apparent disadvantage is overcome by admitting some of the water at the sides of the cone instead of through the throat at the base and so preventing a high velocity at the throat.

One objection raised against the Chance process is the loss of sand and the wear and tear to the pumping machinery by the sand. In this country there are ample supplies of sand, and the cost of replacing the sand should not prove to be excessive. The loss of sand amounts to 2 to 3 lb. per ton of washed coal, the bulk of the loss being from the water-settling tanks. At Mount Union 216 tons of sand were lost in the washing of 165,000 tons of coal during a period of four months. This is equivalent to 2.6 lb. per ton (2,000 lb.) of coal, or 2·9 lb. per ton of 2,240 lb. With sand at 12*s.* per ton, this would be equivalent in England to a cost for sand of $\frac{1}{6}$*d.* per ton of coal washed.

The wear and tear of the sand pump, the water pump and the refuse-discharge valves may be reckoned as equivalent to the cost of renewing the sand pump every six months, and the impellor of the water pump every year. This would cost under $\frac{1}{2}$*d.* per ton of coal treated.

In connection with the question of cost, it should be observed that, in washing bituminous coal, the amount of sand in circulation is very considerably less than the corresponding amount in washing anthracite, and there is, in consequence, far less wear to the pumps. Further, in anthracite washing, the proportion of sand in the separating medium is greater and a greater loss of sand in the products is sustained.

Fig. 169 is a photograph of an experimental model of the Chance washer which has been installed at Lehigh University. A similar one has been erected at the University of Illinois. Its diameter at the top is 20 in., and it can be used for washing either anthracite or bituminous coal. In the photograph the two refuse-removal valves are clearly seen near to the bottom of the picture. The throat is visible as a vertical tube behind the water inlet pipes, of which the uppermost ones are for adding extra water to the mass in the body of the cone.

THE CONKLIN PROCESS

The Conklin process depends upon the same principle as the Chance process, but magnetite is suspended in the water instead of sand to produce a fluid mixture of high specific gravity. Magnetite has the advantage over sand that its specific gravity is higher (5·0 against 2·6), and for this reason the mixture contains a lower proportion of solid material. Because of its higher specific gravity, however, the solid particles have a greater tendency to separate out from suspension, and must, therefore, be used in a finer state of

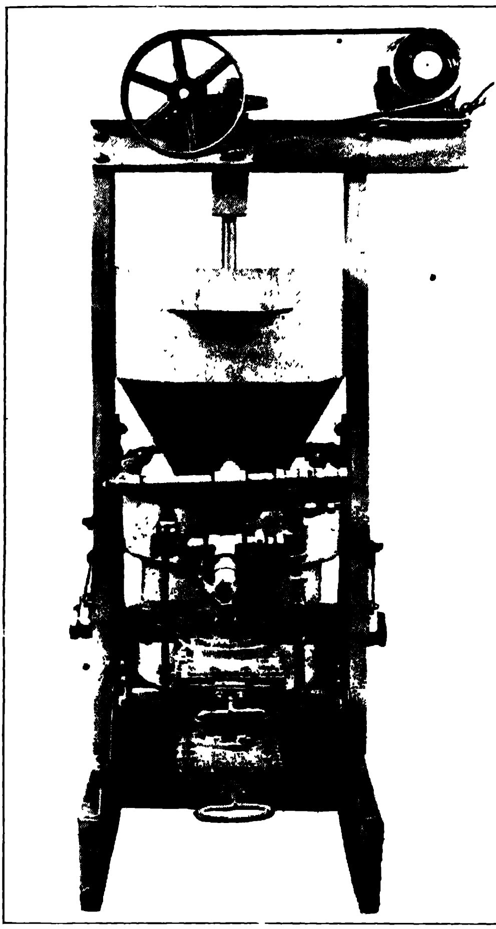

FIG. 169. Experimental Model of Chance Washer.

[To face page 390

division. Actually, magnetite passing through a 200-mesh sieve is employed.

Sections and a plan of a Conklin plant erected in the anthracite district in Pennsylvania are given in Fig. 170. The lay-out of the plant is shown in Fig. 171. The washer consisted of a tank with vertical sides but inclined ends. The coal was fed by a rotating paddle which caused it to become submerged. The heavy particles sank through the fluid medium to a worm conveyor running along the base of the tank, which delivered them to a drainage elevator.

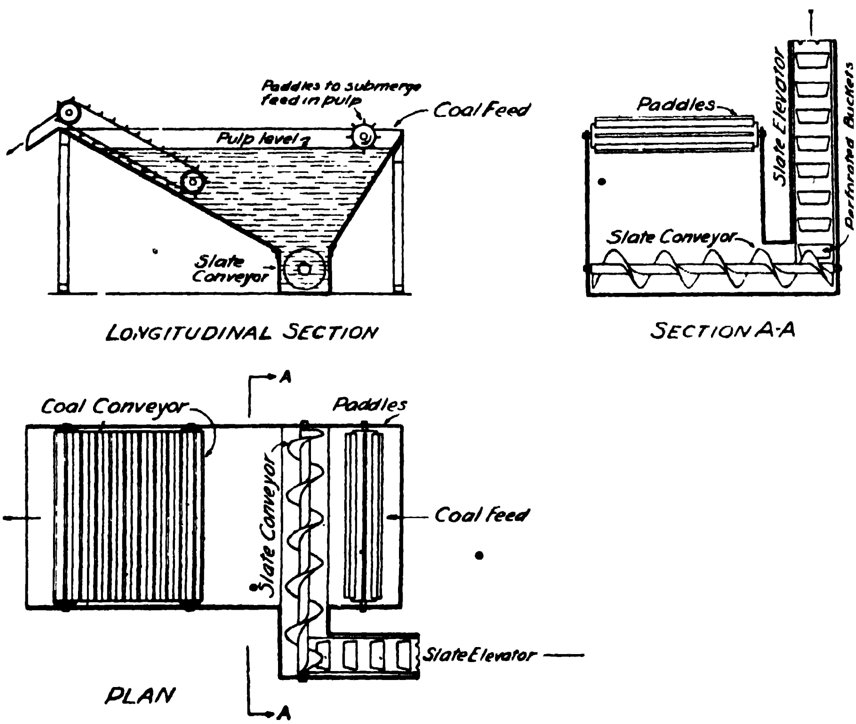

FIG. 170.—Conklin Separator.

The lighter coal particles, however, passed across the tank at or near the surface of the fluid and were removed from the tank by an endless paddle kept in continuous motion.

The sides sloped at angles of 30° at the feed side and 60° at the coal-discharge side; the tank was 3 ft. wide and 6 ft. long at the top, tapering to a trough containing a 12 in. diameter worm conveyor.

The bulk of the magnetite removed with the coal and refuse was recovered by the use of a Dorr classifier and Dorr thickener. In the classifier, all the magnetite overflowed, together with some of the finest coal and shale, and the final separation was effected in the thickener, where the coal and dirt overflowed. Despite these elaborate precautions, 35 lb. of magnetite were lost per ton of coal treated,

a loss which, it is said, could be reduced to 20 lb. Assuming a cost of 5 dollars per ton for magnetite, the loss amountsto 5 cents ($2\frac{1}{2}d.$) per 2,000 lb. of coal treated.

In the Chance process the mixture is continuously agitated by a revolving stirrer and by the current of make-up water supplied at the base of the cone. In the Conklin process no such agitation was

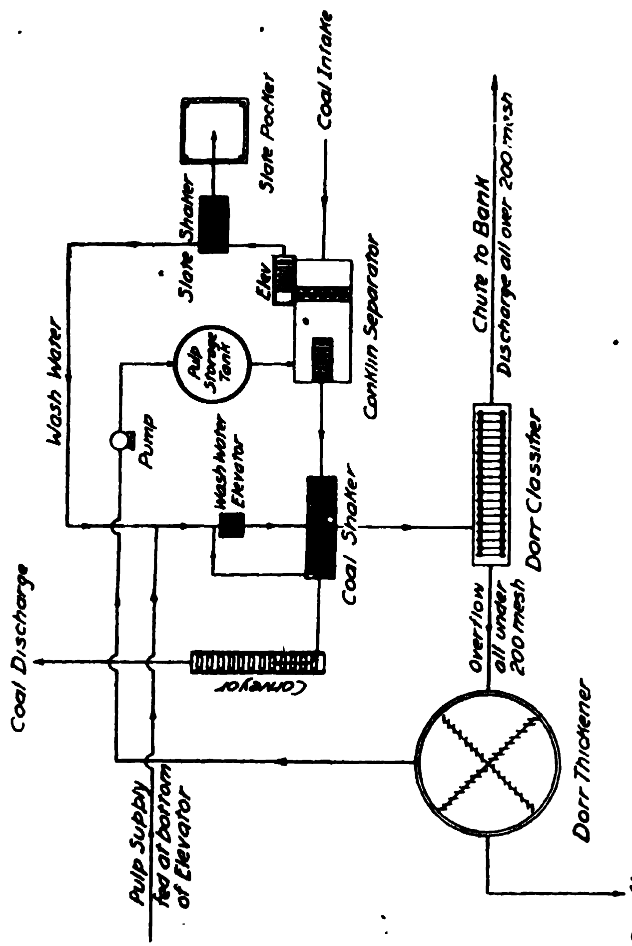

FIG 171.—Flow Sheet of Test Plant (Conklin Process).

necessary, the disturbance caused by the feed paddle and the clean coal remover being adequate. Make-up water and magnetite were added separately and continuously.

In comparison with jigs treating the same coal, the process enabled a greater amount of heavy shale to be removed and a higher proportion of intermediate particles (bone coal) to be recovered. The loss of coal in the refuse was about the same (2 per cent.) in the jigs as in the Conklin plant.

The results were as follows, in each test the size treated being chestnut ($1\frac{1}{4}$ to $\frac{3}{4}$ in.) :—

	Conklin.	Jigs.
Tons per hour	22·3	12·5
Shale in cleaned coal (per cent.) .	1·97	3·72
Bone in cleaned coal ,, ,, .	3·51	1·96
Coal in refuse ,, ,, .	1·97	2·03

So far as can be seen, the only possible advantage of the Conklin process over the Chance process is that it permits of the treatment of smaller particles. The cost of its operation is, however, much greater, and the results published do not suggest that it would be likely to achieve the same degree of efficiency, especially since the figures given were on a closely-sized feed. This small plant, which was largely of an experimental nature, is the only one to have been erected.

THE FRAZER AND YANCEY PROCESS

A dry-cleaning process invented by Professor Frazer and Dr. Yancey, in which a mixture of sand and air with an effective density of 1·45 is used to separate coal and dirt, has been described in Chapter XVII.

THE CLEAN COAL COMPANY'S PROCESS

A process has recently been devised by the Clean Coal Company, in which a solution of calcium chloride (or other cheap salt) is used to float the coal and allow the refuse to sink. By this means theoretically perfect separation of the material floating at a specific gravity of 1·4 from heavier particles may be effected.

The solution is contained in a vessel whose base is tapered. The raw coal is fed into the body of the separating medium by a screw conveyor, and the floating coal is removed by an endless scraper into a vertical cylinder. The refuse is collected from the tapered bottom by an elevator and discharged into a similar, but smaller, vertical cylinder.

The solution adhering to the particles is allowed to drain through grids at the base of each cylinder and the residual salt is washed away by water. The water is added to the top of the draining cylinders in a gentle stream, and the washing continues until the drainings are free from chloride. The success of the drainage depends upon the fact that the calcium chloride solution and the water are not allowed to mix. The water rests as a layer on the heavy chloride solution and the layer gradually descends through the mass with a minimum

amount of diffusion. A certain quantity of weak liquor is, of course, made during washing, and this is collected, evaporated and re-used.

The successful accomplishment of the washing is shown by the fact that only about 30 litres of weak liquor is said to be made per ton of coal washed, and the average loss of calcium chloride amounts to only 2 litres of concentrated solution per ton.

Before the raw coal is fed to the separating vessel, the finest dust is removed. By this means the drainage of the final traces of water is facilitated, and, when the coal is withdrawn from the drainers (through valves at the bottom), it contains less than 8 per cent. of moisture. This is shown by the following results obtained on an experimental unit capable of treating 3 to 4 tons per hour. The results also give the ash contents of the raw coal, the clean coal, and the refuse.

Type of Coal.	Raw Coal. Ash per cent.	Clean Coal.		Refuse. Ash per cent.
		Ash per cent.	Free Moisture per cent.	
Northumberland boiler slack	21·9	2·8	4·6	60·1
Durham smalls . . .	12·9	2·8	3·9	62·0
Durham boiler duff . .	11·7	3·3	5·8	53·5
Durham coking slack . .	15·3	2·9	3·8	63·3
Yorkshire coking slack .	14·6	0·9	6·1	70·0
Welsh anthracite duff . .	16·1	2·0	4·8	57·6

The removal of dust before cleaning has the additional advantage that the fusain and fine dirt particles are not wetted and that less slurry is formed. Dust removal being an essential part of the process, a special form of separator has been devised, in which an air current is passed upwards through the falling coal and the fine particles are removed by elutriation. The efficiency of dust removal is greater than is possible with any form of screen.

The process is still in the experimental stages, but is apparently efficient and economical on a semi-industrial scale.

CHAPTER XXI

FROTH FLOTATION : THEORY

The chief interest in froth flotation processes for the cleaning of coal lies in the fact that they are applicable to the smallest sizes of coal which, as is well known, are often imperfectly cleaned by methods depending principally upon density differences. The separation of sulphide ore from earthy gangue, or of coal from shale, is almost independent of the mass of the particles, since it is based on the differences of their surface properties, for example, their wettability by oil and water. Like most other methods of coal cleaning, froth flotation methods were first applied in ore-dressing practice, and, in recent years, they have attained such an importance that they have largely superseded jigs and other appliances for the concentration of slimes and the smaller sizes of metalliferous ores. Froth-flotation methods are, however, not so suitable for the preparation of coal as for the concentration of metalliferous ores on account of the difficulty and expense of dewatering the cleaned coal. The more valuable ore may be dried by heat at a relatively low percentage cost. Nevertheless, flotation has been successfully and profitably applied to the cleaning of coal in a number of plants, and the principles and practice of flotation are, therefore, worthy of careful consideration, particularly in plants where much slurry is produced.

HISTORICAL DEVELOPMENT

The first suggestion to take advantage of the different wettabilities of ore and gangue by oil was made by Haynes, of Holywell, Wales, in 1860. In his process, which was intermittent, air flotation was not applied. In 1885, Bradford, in the United States of America, recovered sulphides from " tailings " by passing them in a thin stream along an inclined channel on to a water surface ; the gangue, which was readily wetted by water, sank, and the sulphide material, being less readily wetted by water, floated on the surface and was removed at an overflow. This process was a true or simple flotation process as distinct from the modern processes which have been developed from it, and which are froth flotation processes, flotation being secured by the stable attachment of gaseous bubbles to solid particles which are thus rendered buoyant.

The first process resembling froth flotation may be attributed to a woman metallurgist, Carrie Everson, of Chicago, who, also in the year 1885, improved Haynes' process by using sulphuric acid

to enhance the selective wetting of gangue by water. The gaseous effervescence produced by the action of acids on certain ores appears to have led to the discovery of the floatability of oil-filmed particles by aeration, upon which commercial processes of froth flotation now depend.

In 1894, Robson, at the Glasdir Mine, Dolgelly, Wales, agitated ground chalcopyritic ore in a vessel containing water through which a stream of thin oil (made from a thick fatty oil dissolved in paraffin or turpentine) flowed upwards. The upward stream of oil entrained sulphide particles to the water surface, whilst the gangue settled in the vessel. It is said that Robson used as much as 3 tons of oil per ton of ore (Pickard, *Can. Min. Inst. Bull.*, 1916).

The first commercially successful flotation process, however, appears to have been devised by Francis E. Elmore, in 1898, at the Glasdir Mine. Elmore, with his brother, developed the "bulk oil" process from Robson's earlier experiments at the same mine, the success being in a large measure due to the engineering skill of the Elmore brothers in developing a suitable appliance for the process. The ore was crushed in water before the wetted particles were mixed with a thick mineral oil. The mixing was performed in a cylinder, which was rotated slowly to ensure thorough contact between the oil and the particles. The pulp passed from the rotating cylinder into a spitzkasten. The oil was able to displace the water from the surface of the ore, which then floated on the surface and overflowed from the spitzkasten into a launder. The water-wetted gangue, being unaffected by the oil, settled in the spitzkasten and was removed from the base. About equal parts of oil and ore were used, but, after the recovery of some of the oil by centrifuging, about 1 per cent. of oil (calculated on the weight of the ore) was actually consumed in the process, the remainder being returned for re-use. The results of this process were most satisfactory, and the success drew attention to the possibilities of flotation in ore concentration.

At the Broken Hill mines, Australia, it was found to be impossible to recover the zinc from the middlings obtained by water concentration, owing to the large amount of gangue present. After the possibilities of flotation had been demonstrated by Elmore, an attempt was made to recover the valuable ore by floating it, instead of allowing it to sink, as in the water-concentrating (gravity) methods. Potter, in 1902, evolved a flotation process in which the middlings were introduced into a vessel containing dilute sulphuric acid maintained at a concentration of 2½ per cent. of acid and at a temperature approaching the boiling-point. By the action of the acid on the calcite in the gangue, carbon dioxide was evolved, bubbles of which, on stirring, adhered to the sulphide ore and carried it to the surface whilst the gangue settled to the bottom.

Fromont, later in the same year, used gas bubbles generated in a similar way to increase the buoyancy of particles already treated

with oil. He also introduced the use of violent agitation; in his case, to obtain more rapid production of gas.

De Bavay, of Melbourne, in 1904, improved the simple flotation process of Bradford (1885) by allowing particles of a crushed ore to pass from a corrugated conical feed device on to a water surface where the sulphide ore particles floated to an overflow, whilst the gangue sank to the bottom of the vessel. This process, modified by subsidiary aeration and small oil additions, was markedly successful at Broken Hill, except when slimes were used.

In 1904, Elmore brought out a second process, which employed the selective action of oil for sulphide ores and incorporated the idea of floating the oiled sulphides by means of gas bubbles to form a froth at the surface. Aeration was obtained by reducing the pressure above the water surface, so that the air dissolved in the water was released as a number of small bubbles. The bubbles rose to the surface, carrying particles of oiled ore with them.

The pulp was acidulated and the carbon dioxide produced also contributed to the flotation of the ore. The amount of oil used was reduced to 0·5 per cent., and later to as little as 0·15 per cent.

In 1905, Minerals Separation Ltd. took out their basic patent (B.P. 2,803 and U.S. 835,120) in the names of Sulman, Picard and Ballot. The Minerals Separation process was discovered whilst trying out the Cattermole process. In this process, introduced in 1902, and adopted in 1903 by the Minerals Separation Company (which was formed for this purpose), the sulphides were made to sink and the gangue was assisted to rise in an upward current of water. About 3 per cent. of oil and 2 per cent. of soap were added as an emulsion to obtain an agglomeration of flocculent sulphide particles, which, being heavily oiled, stuck to each other in groups or granules and sank to the bottom. The pulp was violently agitated, but a considerable proportion of the mineral was floated by the froth incidental to violent mixing in the presence of air. The agitation-froth process appears, therefore, to have been discovered almost accidentally. More froth was made by using less oil, and a greater proportion of ore was floated, and the omission of the upward current allowed the gangue to sink. Frothing and floating proved to be a better method for recovery of the sulphides than granulating and sinking them. The oil used was decreased to 0·15–0·2 per cent. and the violence of agitation was increased. The sulphide ore particles were attached to air bubbles, which rose to the surface and collected in a dense froth.

Higgins, in 1908, suggested the use of phenols as froth-forming agents, and, later, alcohols, amyl acetate, camphor, eucalyptus oils and turpentine were used for the same purpose.

From these processes modern flotation practice has been developed. Although the Elmore vacuum process was used from 1908 to 1910 to treat many hundreds of thousands of tons of zinc middlings at Broken Hill, it is said that it could not deal with sands and slimes

as effectively as the process patented by Sulman, Picard and Ballot. With the Minerals Separation process, the problem of recovering the zinc at Broken Hill was solved. The fine copper ores in New South Wales, and particularly in America, were successfully treated, and by 1914 the process had become successfully established. Edser (1921) records that over 70,000,000 tons of ore were treated annually by this process.

It was not until 1920 that the experimental work of Bury, Broadbridge and Hutchinson (*Trans.* Inst. Min. Eng., 1920, 60, 243) directed attention to the possibilities of the use of froth flotation for coal cleaning. The first froth-flotation plants (Minerals Separation) for coal cleaning were erected during the same year in Spain and in France. The first British plant was erected in 1922, and, during the following year, plants were erected in Germany and Belgium.

THEORETICAL CONSIDERATIONS *

To appreciate fully the principles of simple flotation and froth flotation it is necessary to consider the properties of surfaces which give rise to the phenomena of surface tension, differential wetting, adsorption, flocculation and other phenomena with a bearing on flotation practice. Each of these phenomena is a surface effect and no present explanation of them is satisfactory except on a basis of the molecular structure of matter.

Simple flotation depends essentially on the different wettabilities of ore and gangue, or of coal and dirt, by water. The wettability of a solid by a liquid, such as water, is indicated more or less quantitatively by the angle of contact made between the free (or upper boundary) surface of the liquid and of the solid. The contact angle is a definite property of the surface tensions of the solid and of the liquid and of their common surface or interface. The interfacial tension between a solid and a liquid may, however, be modified by the addition of minute quantities of certain reagents, and the floatability of different materials may, therefore, be controlled. The "*film*" flotation of a solid at a free liquid surface, accomplished by these means, is the first necessity of successful flotation practice, and the formation of a froth is a subsidiary practical modification.

According to the molecular theory of matter, solids, liquids and gases are composed of finite particles, atoms or molecules, separated from each other by distances which are relatively small in solids, somewhat greater in liquids, and considerably greater in gases All molecules are conceived to be in rapid motion, which increases with the rise of temperature, but which ceases entirely at absolute zero (– 273° C.). At 0° C. the average velocities of typical molecules

* The following treatment of the principles of froth flotation is based on two papers, "A Contribution to the Study of Flotation," by H. L. Sulman, *Trans.* Inst. Min. Met., 1919; and "The Concentration of Ores by Flotation," by E. Edser, Fourth Report on Colloid Chemistry, London, 1922.

in the gaseous, liquid and solid states are computed to be: gaseous hydrogen, about one mile per second; liquid water, about one-third of a mile per second; and solid silica, about one-quarter of a mile per second. The velocities of molecules of all substances, at equal temperatures, are inversely proportional to the square roots of their molecular weights. The average distance travelled by any molecule before collision with other molecules is small in a liquid and still less in a solid. The "mean free path" for a water molecule is supposed to be less than the molecular diameter. The molecular movement gives rise to the mobility of a liquid as a whole, and also accounts for diffusion phenomena.

In a solid the molecules are subject to a greater restraint than in liquids, though the general similarity between the volumes of a substance in the solid and liquid states does not indicate a much closer inter-molecular spacing. The molecules of a solid are considered to oscillate about more or less fixed positions, and to be constantly colliding with their neighbours. The phenomena of crystallisation and magnetism show them to be capable of taking up definite axial directions or orientation, but, being unable to move independently of each other with any readiness, their assemblages possess rigidity and form. Nevertheless, there is much evidence to show that the molecules in a solid can slowly change their relative positions, as in the gradual crystallisation of masses of steel, in which there is a slow segregation and orientation of like molecules. The comparatively sudden volume changes which take place during the heating up of a silica brick coke-oven (due to the inversions of the various forms of tridymite, cristobalite and quartz) provide further evidence of molecular movement in a solid. Further examples may be found in the phenomena of solid diffusion, solution and sublimation. For the theoretical considerations involved in flotation it is important to establish clearly the idea that the molecules in a solid surface are capable of movement and of changing their relative positions, for such movement provides the most plausible explanation of many flotation phenomena.

The molecules of all substances are subject to gravitation and molecular forces. Gravitational forces affect all the molecules in a body, so that they are directly proportional to the masses of the attracting bodies. They vary inversely as the square of the distance between the bodies, and so may act over sensible distances. Gravity is independent of the chemical nature of the substances.

Molecular attractions are much more intense than gravitational forces, but are manifested only over minute distances (of the order of $\frac{1}{10000}$ mm.) and are therefore only sensible between molecules when they are brought comparatively close together. Edser (Fourth Report on Colloid Chemistry, London, 1922) deduces that the attraction varies for different substances inversely as the sixth to the ninth power of the distance; experimental data for liquids indicate that molecular attraction varies inversely as the eighth power of

the distance. There is therefore a steep gradient in the intensity of molecular attractions, even within the limiting distances through which they are exerted. Unlike gravitation, molecular attraction varies with the chemical nature of the molecules. It is these molecular forces of minute range which give rise to the properties of cohesion, adhesion, surface tension, and adsorption, all of which are brought into play in flotation.

Cohesion.—If, in a body of liquid, as in Fig. 172, a single molecule G is considered to exert attractions on all the molecules in a sphere described round it, the attractions are equal about any plane in the sphere and there is therefore no resultant of the molecular forces. The attractions that the molecules exert across any imaginary plane in the liquid constitute cohesion. Consider, however, a molecule E, near the surface CD of the liquid, with a sphere described around it, indicating the limit of its range of molecular attraction ;

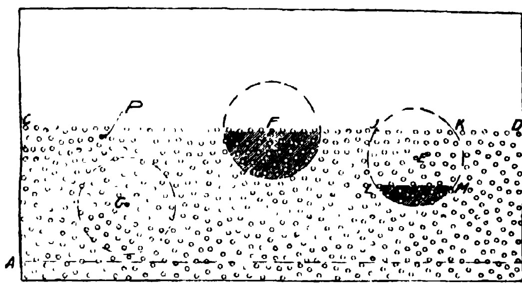

FIG. 172.—Surface Tension : Diagram.

the molecule E will attract all molecules within this spherical range and all other molecules within the sphere will themselves act upon E in a like manner. The sphere of attraction exerted by molecules of the liquid cuts the surface at JK, beyond which there are no reciprocally-attracting like molecules. Within the portion of the circle *JKLM, in the plane of the paper*, the attractions between the molecule E and the other molecules included in the section JKLM are balanced, but the molecules in the shaded portion of the circle below the plane LM exert a pull towards the interior upon the molecule E. Similarly, there is a still greater force attracting the molecule F, situated on the surface, towards the interior of the liquid. The attraction towards the interior of a liquid may be held to increase as the molecule moves away from the centre towards the surface, and to reach its maximum when the molecule is at the surface. If a molecule be moved from the interior to the surface of a liquid, work will be done against the forces of attraction, and consequently, on reaching the surface, the molecule will possess potential energy in excess of that which it possessed in the interior. Thus

molecules at a surface possess greater energy than those in the interior of a body of liquid. The surface of a liquid is also slightly less dense than the interior, and a state of strain is induced.

This simple statement of surface energy is incomplete when considering larger and more complex molecules (such as occur in coal), which do not exert uniform attractions in all directions. Nevertheless, for the present purpose, it is satisfactory, and enables an explanation to be put forward of the principles upon which the froth flotation of coal depends.

Surface Tension.—The latent potential energy in unit area of surface is termed the " surface energy " of the substance, and, since in all circumstances a system tends to arrange itself so that its free potential energy is at the minimum, the free surface bounding a liquid will take up such a form that its area is a minimum. The chief exhibition of this surface energy of a liquid is in the phenomenon of surface tension, which acts as if the liquid were surrounded by a skin always tending to contract uniformly.

Since surface tension results from the unbalanced molecular forces existing at the surface, it should bear some fundamental relationship to the balanced internal molecular forces, for example, to the cohesion. This relationship can be proved to exist, for substances which have low cohesive power possess low surface tension, and *vice versâ* (see Table 105).

Surface tension may be defined as the force equal to the total

TABLE 105.—RELATIONSHIP BETWEEN COHESION AND SURFACE TENSION (EDSER)

Liquid.	Cohesion. Dynes $\times$ 10 ^{9}.	Surface Tension in Dynes per cm.
Ether	1·559	17·1
Hexane	1·713	17·4
Methyl alcohol	1·781	22·0
Ethyl alcohol	1·837	23·2
Octane	1·838	23·3
Carbon tetrachloride	2·237	26·0
Chloroform	2·381	26·9
Xylene	2·378	28·5
Toluene	2·426	29·1
Carbon bisulphide	2·490	31·1
Cresol	3·123	36·8
Water	3·645	38·5*
Mercury	13·17	440

* At 100° C.

molecular attraction exerted across a line 1 cm. in length in the free liquid surface, or, alternatively, as the energy, expressed in ergs, required to create a fresh surface of 1 sq. cm. under isothermal conditions.

Some evidence has already been recorded to illustrate the dynamic properties of the molecules in the surface of solids. Further evidence is to be found in the work of Beilby (Roy. Soc. *Proc.*, 1903, A, 72, 218), who found that, by firmly stroking copper or calcite with the finger tips covered with chamois leather, a surface layer, estimated as being 1,000 $\mu\mu$ in depth, flows under the pressure like a fluid. The "fluid" layer retains its mobility for but a short period, and resolidifies into a vitreous amorphous mass harder than the original crystalline surface, and more soluble in acids. By dissolving away this layer, furrows resulting from the initial friction become visible under a high-power microscope at a computed depth of probably more than 1,000 molecules.

Other evidence of a similar nature could be recorded to strengthen the view that the surface of solids must often be not only granular, but also plastic, and thus frequently capable of penetration by the molecules of gases, liquids, and even of other solids. Most solid/liquid interfaces in flotation practice might be conceived to have the molecules of the liquid interlocked or "rooted" into those of the solid at the plane of contact.

The surface tension of liquids may be readily measured; for example, by weighing the pull exerted on a special balance pan by a film of the liquid. The direct determination of the surface tension of solids, however, is not susceptible of direct measurement, but indirect methods may be used as a guide to the actual values. The principle of the determination of the surface tension of a liquid may be visualised by imagining a liquid separated into two parts at a plane in its interior. Work is done in effecting the separation because of the attraction across the plane where the liquid is divided into its two parts. If W denotes the work done per unit area of the plane in overcoming the attraction exerted across it by the two parts of the liquid, then $W = 2S$, where S is the surface tension of the liquid. Using the same reasoning, the ease with which two free surfaces may be produced in a solid by cleavage is an indication of the surface tension of a solid. Thus a friable substance like coal would, in general, possess a lower surface tension than, say, pyrites.

When a solid can be melted, its surface tension in the molten state can be measured, and this may give some indication of the surface tension in the solid state. Edser (*loc. cit.*) has also shown that the surface tension of solids may be calculated from certain other physical characteristics. For example, he calculates that the surface tension of quartz is 920 dynes per centimetre, and of iron pyrites, 1,175 dynes per centimetre. In general, it would appear that the surface tensions of solids are higher than those of liquids, in conformity with their greater forces of cohesion.

Adsorption.—Since, according to Edser (*loc. cit.*), the molecular attraction of liquids varies inversely as the eighth power of the distance, it may be shown that about 95 per cent. of the energy corresponding to the surface tension of a liquid is localised in the extreme surface layer one molecular diameter in thickness. It is within this surface layer that molecules foreign to the pure liquid tend to be concentrated. This process of surface concentration of impurities is known as adsorption.

Interfacial Tension.—The surface tension of a solid refers to a surface in contact with air. When, however, a second solid possessing a different value for surface tension is placed on the first solid, a different situation arises, for the molecules in each surface exert some attraction across the common surface (the interface). This molecular attraction across an interface between two dissimilar substances is called *adhesion*, and is measured by the force per unit of area necessary to separate the substances. When the interface is formed between a solid and a liquid the phenomenon of adhesion is termed *wetting*. The molecular attraction between like molecules (cohesion) on either side of the interface is greater than the molecular attraction exerted across the interface between dissimilar molecules (adhesion), and the resultant unbalanced forces give rise to *interfacial tension.* When the interface is formed by the contact between a liquid and a solid, interfacial tension has an effect in resisting lateral extension of the interface. The forces involved may be equated, using the symbol σ for surface tension :—

Let σ_1 represent the surface tension of the liquid (or the work done in creating 1 sq. cm. of free liquid surface) ;

σ_2 the surface tension of the solid (or the work done in creating 1 sq. cm. of free solid surface) ;

σ_{12} the surface tension of the liquid/solid interface (or the work done in creating 1 sq. cm. of solid/liquid interface).

Let W_{12} be the work done in separating the interface at a plane of 1 sq. cm. area, into two fresh surfaces of 1 sq. cm. each.

Imagine the solid and liquid surfaces to be brought together. If no work is done in separating them the total energy of the interface (interfacial tension) will be the sum of the energies of the two surfaces, or

$$\sigma_{12} = \sigma_1 + \sigma_2 \text{ when } W_{12} = 0.$$

In this case the molecules of the liquid would exert no attraction across the interface upon the molecules of the solid ; nor would the molecules of the solid attract the molecules of the liquid, and no adhesion or wetting would arise. The strain in the system is unrelieved and the interfacial tension is, therefore, at a maximum. The case is hypothetical, for no example of zero adhesion (absolute non-wetting) is known ; should a case of complete non-wetting occur, the liquid would form complete spheres resting on the solid, and there would be no interface.

The relation between the interfacial tensions for the phases, solid/air, liquid/air, and solid/liquid may best be appreciated by considering an air bubble in contact with the interface between a solid and a liquid (water) as in Fig. 173. A contact angle θ is made between the air/water and the water/solid surfaces. At a point, P, where the three phases meet, equilibrium will be established by the algebraic sum of the three tensions, namely, the tension of the solid, σ_2, the component of the tension of the liquid resolved parallel to the surface, $\sigma_1 \cos \theta$, and the interfacial tension, σ_{12}.

Then $$\sigma_2 = \sigma_1 \cos \theta + \sigma_{12},$$

or $$\frac{\sigma_2 - \sigma_{12}}{\sigma_1} = \cos \theta.$$

This relationship may be stated in general terms: that the extent to which the surface tension of the solid is reduced by contact with the liquid determines the angle of contact between the air/liquid and liquid/solid interfaces. The greater the molecular attraction (or adhesion) across the interface between the solid and the liquid, the more relieved are the surface strains, and the more readily can the solid be wetted. The angle of contact is a measure of the interfacial tension and, therefore, of the degree of adhesion. It is, therefore, also a measure of the ease of wetting of the solid by the liquid.

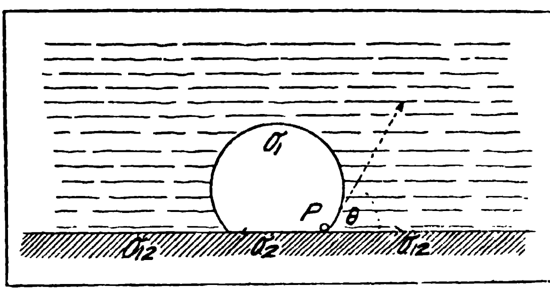

FIG. 173 — Diagram of Contact Angle.

The relationship between the contact angle, the degree of wetting, and the various tensions involved may be examined in the following examples (due to Sulman, *loc. cit.*). In each example a small air bubble is considered to be in contact with a liquid/solid interface, and a small drop of liquid on a solid surface is regarded as being in contact with air.

(1) Fig. 174. (Theoretical condition of non-wetting.)

In this case, θ = 180 degrees and $\cos \theta = -1$,

whence $$\frac{\sigma_2 - \sigma_{12}}{\sigma_1} = -1,$$

or $$\sigma_{12} = \sigma_2 + \sigma_1.$$

Since the sum of the surface energies of the solid and liquid phases apart is $\sigma_2 + \sigma_1$, no reduction of energy has taken place in this example, and the interfacial tension is at its maximum ; the degree of adhesion is therefore nil and no wetting occurs. A small drop of liquid would form a complete sphere on the surface of the solid. This case is hypothetical, for no contact angle of 180 degrees is known for a solid/liquid interface (the highest known value being 148 degrees for mercury/glass). With some pairs of immiscible liquids

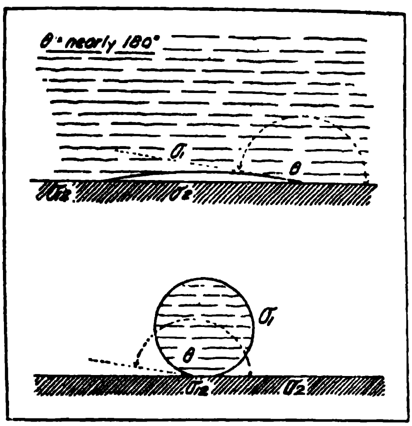

FIG. 174.—Diagram of Contact Angle.

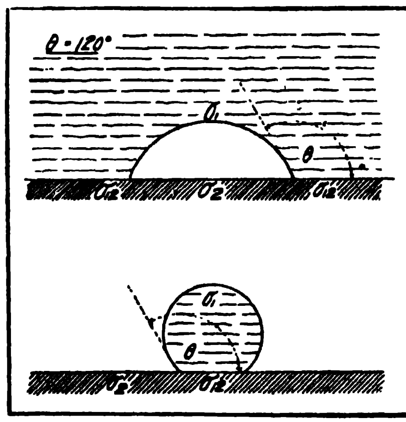

FIG. 175.—Diagram of Contact Angle.

higher contact angles are known, but in all solid/liquid interfaces some degree of adhesion (wetting) occurs.

(2) Fig. 175.

In this case, $\theta = 120$ degrees, $\cos\theta = -\frac{1}{2}$,

whence $$\frac{\sigma_2 - \sigma_{12}}{\sigma_1} = -\tfrac{1}{2},$$

or $$\sigma_{12} = \sigma_2 + \tfrac{1}{2}\sigma_1.$$

In these circumstances, the interfacial tension is reduced by $\frac{1}{2}\sigma_1$, and some degree of wetting occurs.

(3) Fig. 176.

In this case, $\theta = 90$ degrees, $\cos\theta = 0$,

whence $$\frac{\sigma_2 - \sigma_{12}}{\sigma_1} = 0,$$

or $$\sigma_{12} = \sigma_2.$$

Under this condition interfacial tension is reduced to the value of the solid tension and only partial wetting occurs. An air bubble would be stable at a solid/liquid surface. The contact angle considered in this example (90 degrees) is approximately the maximum contact angle obtained with a number of mineral ores (iron pyrites,

calcite, marcasite) and water. For easy flotability a contact angle approaching (or preferably exceeding) 90 degrees is required.

(4) Fig. 177.

In this case, $\theta = 60$ degrees, $\cos\theta = \frac{1}{2}$,

whence $$\frac{\sigma_2 - \sigma_{12}}{\sigma_1} = \tfrac{1}{2},$$

or $$\sigma_{12} = \sigma_2 - \tfrac{1}{2}\sigma_1.$$

When $\theta = 60$ degrees, the interfacial tension is less than the surface tension of the solid, and the extent of wetting is greater than in the previous examples. A relatively strong external force would still be necessary to dislodge a bubble from a solid/liquid interface, so

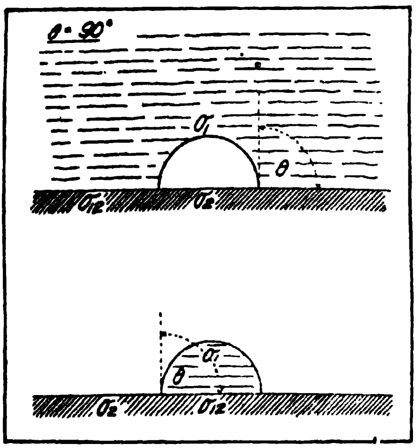

FIG. 176.—Diagram of Contact Angle.

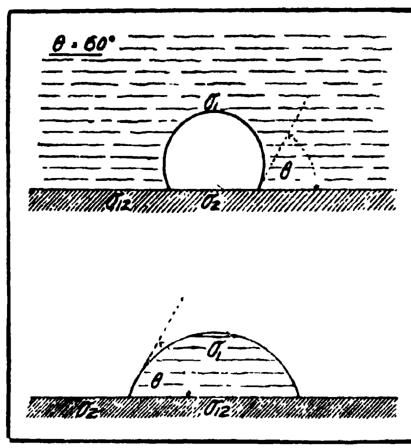

FIG. 177.—Diagram of Contact Angle.

that flotation by air bubbles would be possible, but less easy than with a larger angle of contact.

(5) Fig. 178.

In this case, $\theta = 0$, $\cos\theta = 1$,

whence $$\frac{\sigma_2 - \sigma_{12}}{\sigma_1} = 1,$$

or $$\sigma_{12} = \sigma_2 - \sigma_1.$$

With an angle of contact of zero, a drop of liquid would spread completely over the surface of the solid ; an air bubble at the liquid/solid surface would only rest tangentially on the surface and would leave the solid surface with the application of the least external force. Even when the contact angle is greater than 0 degree, but is still small, flotation by aeration is impossible, owing to the instability of the bubble on agitation.

In example (5), in spite of a zero contact angle, a residual surface

energy will usually obtain, for the solid tension σ_2 is usually greater than the liquid tension σ_1. Sulman has consequently called this condition the "angular limit of wetting," and remarks that complete wetting, when no residual energy remains ($\sigma_2 = \sigma_1$), occurs when particles of solid, if not too large, become true suspensions in a liquid. This condition is called "complete deflocculation."

Flocculation.—When a powdered ore is mixed with water, agitated, and allowed to stand, it may be observed that the particles in the upper layer of the turpid pulp are agglomerated into clusters which tend to sink. The clusters may be shown to consist of numerous small air bubbles coated with finely-divided ore. On adding a small amount of sodium silicate, however, a uniform turpidity of the pulp will occur in which, as distinct from the previous mode of

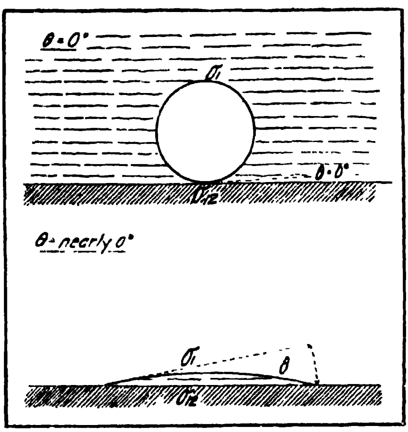

FIG. 178.—Diagram of Contact Angle.

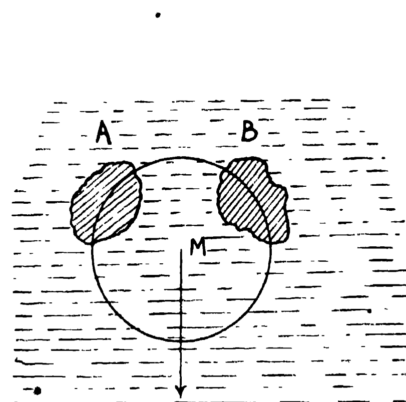

FIG. 179.—Flocculation Diagram.

aggregation, each particle is separated from its neighbours. This suspension may persist for several days. The first condition, of aggregation or agglomeration of particles, is called "flocculation," and the condition of suspension of isolated particles "deflocculation."

Edser (*loc. cit.*) advances an explanation of these phenomena on molecular grounds. Let A and B, Fig. 179, be two particles in a liquid in which M represents one molecule and the circle its radius of molecular attraction. If this sphere encloses parts of the particles, A and B, the molecule, M, is not necessarily in equilibrium, as it would be if the particles were absent. In the absence of the particles, A and B, the space they occupy would be filled with liquid, and the molecular forces acting on the molecule, M, would probably be either greater or less than they are in the presence of the particles. If the particles attract M with a force less than that which would be exerted if the spaces they occupy were filled with liquid, the mole-

cule, M, will be pulled away from the space between the particles; similar forces will act on all the molecules of liquid between the particles so that the liquid will be sucked out from between them, and they will appear to attract each other. Similar reasoning can be advanced to explain apparent repulsion, or equilibrium. Edser develops a mathematical method to show that the condition for apparent attraction (which is simply flocculation) is satisfied when a finite contact angle is formed, and, as this is also the condition for flotation, it follows that all materials that can be flocculated can also be floated. Edser also shows that the effect of air bubbles in promoting flocculation is very marked (as in the flocculation of a precipitate of silver chloride in chemical analysis, by shaking), and argues that the mechanical action of the air bubbles will bring together particles which are too widely dispersed to exert any attraction on each other.

Differential Wetting.—In general, oils make lower contact angles than water with metals, metallic sulphides, graphite and coal. Oils therefore wet such surfaces more easily than water. In a system containing a solid (coal) and two liquids (oil and water), since oil would reduce the interfacial tension by a greater amount than water, the oil would wet the solid preferentially. On the other hand, the interfacial tension between gangue (or shale) and water would be less than that between gangue and oil, and water would wet the gangue more readily.

Although an exact relationship between chemical properties and wetting power or wettability cannot yet be stated, a guide may be found in similarity of physical behaviour. Tilden (Chemical Philosophy, London, 1912, p. 331) has pointed out the broad parallelism in composition which exists between substances markedly soluble in each other. For example, petroleum is a better solvent than is alcohol for paraffin wax, but alcohol is a better solvent than liquid hydrocarbons for resins, which are oxygenated bodies; carbon bisulphide is the best solvent for sulphur, and water is the best solvent for most oxygenated salts. Since complete wetting precedes solution, the same principle holds for wetting power as for solution. Thus it is found that water readily wets oxides, hydroxides, quartz, silicates, and oxygenated inorganic salts, but does not so readily wet hydrocarbons, waxes, sulphur, diamond, graphite and many metals. On the other hand, oils readily wet sulphur, sulphide ores, metals, graphite and coal, but do not readily wet oxides, quartz or oxygenated inorganic salts.

Hysteresis of Contact Angles.—When a contact angle is measured whilst a liquid is advancing over a dry surface, a certain value, θ_1, is obtained. If the contact angle is measured as the liquid is receding from the wetted surface, a lower value, θ_2, is obtained. It has been suggested by Sulman that this difference between the

maximum and minimum values of the contact angle should be called the hysteresis of the contact angle. The existence of the two limiting contact angles may be observed on placing a drop of water on a clean, dry piece of glass and then tilting the glass until the drop begins to flow. The two angles of contact may be observed, one at the front of the drop as it advances, and one at the back as it recedes. Edser (*loc. cit.*) observes that, in this example, the surface of the solid from which the liquid has just receded has a higher surface tension than that of the original dry surface ; as adsorption of water would reduce the surface tension of the solid, he suggests that water is absorbed. The contact angles for several substances with water is recorded in Table 106.

TABLE 106.—CONTACT ANGLES WITH WATER

Substance.	Contact Angles.		Hysteresis.
	θ_1 (max.).	θ_2 (min.).	$\theta_1 - \theta_2$.
Marcasite	83·5	56·5	28·0
Iron pyrites	87·0	25·5	61·5
Quartz	58·5	19·5	39·0
Calcite	85·5	39·6	45·9
Glass	39·5	33·0	6·5

Hysteresis of contact angle is of considerable importance in flotation, for it imparts a wide range of equilibrium to a particle floating by surface tension. With only one definite value, a slight disturbance would cause the particle to sink, whereas with a high hysteresis value greater stability is ensured.

Alteration in Surface Energy of Solids.—When a solid is fractured, two new surfaces are exposed. The arrangement of the molecules in the new surfaces is the same as in the interior of the solid immediately after fracture, for a certain interval of time must elapse before the molecules near or at the new surfaces can take up their new and wider spacing, which, as previously considered, is due to the resultant molecular force of attraction acting towards the centre of any particle. Because of this interval of time, the surface tension of freshly-broken surfaces is lower than that of surfaces which are not freshly exposed and have attained equilibrium. If a liquid is brought into contact with a freshly-exposed surface, the molecular forces causing adhesion or wetting are greater than in an old surface. In other words, wetting of freshly-formed surfaces is easier. This has the important practical result that, in wet crushing, surfaces formed under water are readily wetted, and gangue, or

shale, will tend to remain well wetted by water throughout subsequent operations.

An alternative hypothesis advanced to explain flotation, hysteresis, and the "ageing" of surfaces of minerals is that a film of air is deposited on them. This theory has been discredited by Edser, who showed that air was not condensed to a measurable extent upon the surface of minerals exhibiting these phenomena.

Adsorption.—It has already been stated that when a pure liquid is diluted with a small quantity of a second liquid, the molecules of the impurity tend to concentrate in the surface layer, a phenomenon known as positive adsorption. The presence of foreign molecules, whether solid or liquid, in the surface layer of a liquid, diminishes its surface tension, since they reduce surface strain. Substances which show the most positive adsorptions are those of heavy complex molecules, for example, tannic acid, soap and colloidal substances generally, including coal. Finely-divided mineral sulphide particles are positively adsorbed at a water surface.

When two substances are dissolved in one solvent, the more strongly adsorbed of the dissolved substances tends to displace the less strongly adsorbed from the surface layer. For example, the detergent properties of soap are due to the power of soap to displace molecules of grease from a solid surface. The phenomenon of selective adsorption is of great importance in flotation, as will be seen later. Negative adsorption is evidenced when the molecules of a foreign substance are more strongly attracted by the molecules of the pure liquid than by each other, and concentration occurs of foreign molecules in the body rather than in the surface of a liquid, as is the case in solution. The concentration of molecules in the body of the liquid increases the surface tension.

Stability of Films.—In flotation by means of air bubbles, the stability of the liquid films formed round the bubbles of air is of great importance. The stability of the liquid film is ensured by the presence in it of a positively adsorbed substance. Suppose, for example, that a liquid film is stretched and therefore thinned, molecules of the pure liquid from the inside of the film will be drawn into the surface layers and will therefore dilute the molecules of the adsorbed substance in the surface. Because of the dilution of the adsorbed substance, the tension of the surface layer will be increased, thus counteracting its tendency to break as a result of thinning. On the other hand, if the film contracts, the foreign molecules will become more concentrated in the surface layer and the tension of the surface will be reduced. Thus, if the film becomes thinner, the surface tension increases, and if the film becomes thicker, the surface tension is decreased. Consequently, the equilibrium of the liquid film is maintained because the surface tension adjusts itself to counteract the forces causing stretching or contraction.

The essential condition for bubble-stabilisation is "contamination" of a pure liquid, or the presence of foreign molecules in a pure liquid. Bubbles produced in a pure liquid continually unite and form larger bubbles which are less stable. According to Edser (*loc. cit.*), three classes of liquid mixtures favour aeration or stable bubble formation. The first class includes true solutions, in which the solute is generally sparingly soluble in the solvent, but diminishes its surface tension. Solutions of cresol, phenol, camphor, aniline, amyl alcohol and amyl acetate in water are examples of this class. The second class comprises emulsions of substances which are practically insoluble in water—for example, soap, oleic acid, and olive oil. The third class comprises aqueous solutions of inorganic salts which are highly soluble in water—for example, sea-water, which froths very noticeably during tidal phenomena.

Pure organic liquids (such, for example, as the saturated hydrocarbons), are unable to cause aeration of an agitated body of water, because they do not dissolve in water and contaminate the liquid film. They do not, therefore, cause a reduction of the surface tension. They are, nevertheless, good solvents for substances such as oleic acid and cresol, which favour aeration.

Edser advances an explanation of the non-coalescence of bubbles in a liquid to which an aerating agent has been added. From the same reasoning as that advanced to explain flocculation, he argues that, in a pure liquid, the liquid will tend to be sucked out from between two bubbles which can then approach each other and coalesce. When surface contamination has occurred, however, the molecular concentration in the space between the bubbles will be different from its concentration in the bulk of the liquid. When the bubbles are stationary, the osmotic pressure (the equivalent in liquids, of gas pressure in gases) due to the concentration in the space between the bubbles, will be balanced by the molecular forces that produce concentration. When the bubbles begin to approach each other, a change occurs in the molecular forces producing the local concentration and the osmotic pressure becomes unbalanced, the resultant force tending to stop the motion of the bubbles.

When an aerated liquid is allowed to stand, the bubbles rise towards the surface with a speed depending on their size. With an increase in the number of bubbles in a given volume of the liquid, near to its surface, the amount of liquid in the spaces between the bubbles diminishes. With the closest possible packing of equal-sized bubbles, the percentage of interspace filled by liquid is 25 per cent., and each bubble is in contact with twelve others. When the interspace filled with liquid is less than 25 per cent. of the whole volume, the bubbles lose their spherical shape, and, as the interspace is further diminished, the constituent bubbles of the froth formed approximate to a series of similar cells each with twelve plane walls. For a stable bubble formation of this type, each bubble will take the form of a rhombic dodecahedron; at any edge of one facet three

" bubbles " will be in contact, and the angle between any two adjacent facets of the three " bubbles " will be 120 deg., so that the surface tensional forces are in equilibrium.

Even when this condition is complied with, the conditions at the boundary of the froth must be such as to promote stability. Nearly all frothing liquids are able to wet solids and, therefore, to adhere to them. Consequently, a film of the frothing liquid will adhere to the sides of the containing vessel and ensure stability at the sides. If the upper boundary surface of the froth is in contact with the atmosphere, the upper surfaces of the top layer of " bubbles " must assume a curved shape in order to maintain a higher pressure in the interior of each bubble than the atmospheric pressure outside it. Hence the upper boundary surface of a froth is the least stable portion of the boundary, and a froth always begins to break down at the surface in contact with the atmosphere.

When numerous solid particles are present in a froth, its stability is brought about by forces additional to those already considered. It has been shown that finely-divided mineral sulphide particles may be positively adsorbed at a liquid surface or, in other words, are able to reduce the surface tension of water. Each bubble is coated with solid particles, and, when the bubbles come into contact with each other, the attractions between the solid particles increases the stability of the froth. In the practice of flotation, most of the reagents used are incapable of producing a permanent froth in the absence of solid particles, and the stability of the froth produced in practice will depend on the ratio of the solid contents of the froth to the total area of the film. Excessive aeration may produce an enormous area of film surface with a small concentration of solid particles. Such a condition would give rise to a froth which is less stable than the froth produced by a less lavish aeration, in which there would be a greater concentration of solid particles. A froth which is not particularly stable does not, of necessity, break down immediately it is formed, and, in practice, it may be possible to allow the froth to overflow from a flotation cell before it breaks down. In a series of cells the froth may be sufficiently stable in the first few cells but quite unstable in the last, unless different conditions are employed in the later cells.

GENERAL CONSIDERATIONS

It has been shown that simple flotation, without the necessity of a froth, is possible when a material is not readily wetted by water, this being the principle underlying Elmore's bulk-oil process. Film flotation, or flotation on the free liquid surface of water, would necessitate a large expanse of surface for the commercial flotation of a finely-divided ore, and it is to overcome this difficulty in practice that recourse is made to the use of a froth of air bubbles, which, in effect, increases enormously the liquid/air surface for a given ground

space. Edser (*loc. cit.*) calculates that, for a ton of ore mixed with 4 tons of water, the superficial area of the air/water interface is $2 \cdot 4 \times 10^7$ sq. cm., or about $1\frac{3}{8}$ sq. miles, although the ground space used is only of the order of 10 sq. yd.

The practical requirements of flotation are a low wettability (by water) of the material to be floated, and a high wettability (by water) of the material not required to be floated. Reagents are necessary both to induce bubble formation, and to stabilise the froth produced. The materials to be floated may be of sufficiently low wettability in their untreated form, or it may be advisable to add reagents to reduce their wettability by water. Gangue is usually of sufficiently high wettability to be non-floatable, and, if not, its wettability may be increased by wet crushing, or by the addition of suitable "gangue-modifying agents," which tend to deflocculate it. A froth cannot be obtained in pure water, and, under almost all conditions, the addition of a frothing agent is essential, and usually, a froth-stabilising agent as well. A froth-stabilising agent, by being adsorbed at the mineral surface, will also increase its floatability.

In flotation practice with metalliferous ores, reagents are required to discharge three duties, namely: (1) Aeration, or the production of numerous minute air bubbles in the water; (2) stabilisation of the froth produced (simultaneously increasing the floatability of useful mineral particles); and (3) deflocculation of the gangue to render it unfloatable. A single agent may discharge two or more of these duties. For example, sulphuric acid deflocculates quartz, and thus renders it unfloatable, but it increases the floatability of zinc blende; sodium silicate deflocculates most gangue materials, but increases the floatability of chalcopyrite and other copper sulphides. Eucalyptus oil, oleic acid, turpentine and many oils act both as frothing agents and as froth stabilisers. Tannic acid is an excellent aerating agent for water alone, but its value in this respect is modified by the presence of ores, unless it is added in excessive quantities. In such an excess, however, it deflocculates most minerals. Saponin acts in a similar manner. If soap is added to water in large quantities, nothing at all will float, and even sulphur, which is usually the most readily floatable of all substances, becomes unfloatable in a 0·5 per cent. aqueous solution of soap. Soap and gelatine, although they are strong frothing agents for water alone, are so strongly adsorbed by coal that no aeration results in its presence. Sea-water is an excellent aerating agent, and, from this point of view, could be used for coal flotation but for the deleterious effect of salt in coal when used, for example, in coke ovens. Saturated hydrocarbons are quite useless as aerating agents. They are, however, good wetting agents for metallic sulphides and coal, and if a particle of coal is covered with a film of a saturated hydrocarbon, the wettability of the particle by water is reduced. Oleic acid acts as an aerating agent and also reduces the wettability of particles by

water, a property which is enhanced by the addition of small quantities of the lighter petroleum oils.

Amyl-alcohol in low concentration (1 part in 5,000 to 20,000 parts of ore pulp, or from ½ to 2 lb. per ton of ore in 4 tons of water) has been extensively used as a frothing agent, though it has lately been superseded by cresol in similar or slightly higher proportions. The water-soluble portion of several fixed, and many essential, oils serves equally well the same purpose. Since a minute amount of an insoluble oil, adsorbed at the mineral surfaces, will decrease their adhesion for water, a single oil may serve, not only as a frothing agent, but also in enhancing the floatability of metal sulphides. If an oil be used alone it must yield sufficient aqueous solute; it has been found that eucalyptus oil, pine oils, turpentine, the terpenes and their hydroxy-compounds, camphor oils, sulphonated oils and many aliphatic and aromatic compounds, fulfil this requirement. Tar oils containing phenols and cresols are widely used on account of their cheapness.

The agitation of an oil with water may produce an oil and water emulsion. Calcite cleavages, and many clays and slatey gangues, however, adsorb oils—particularly unsaturated oils—when exposed to such emulsions, and, in this circumstance, the addition of gangue-modifying agents is necessary for deflocculation. Such modifying agents include sulphuric acid, caustic soda, sodium carbonate, sodium silicate, silicic acid, and act not only by adsorption, but also by incipient chemical action.

The possibility that over-aeration may reduce the stability of a froth has already been considered. "Overcrowding" of solid particles also has an important result. If the constituents of an ore exhibit different contact angles with water, a sufficiency of particles with high contact angles with water will displace from a froth those particles which make lower contact angles (Sulman, *loc. cit.*). This permits differential flotation without the use of special reagents. The effect is familiar in the multiple-box apparatus used for coal flotation, in which the froth produced in the first box is richest in particles of higher contact angle (usually lowest in ash) and a different type of product may be produced in successive boxes. By reason of this phenomenon, zinc blende may be separated from the galena with which it is often associated. The practice of returning intermediate froths to the head-box of the series accentuates selection, increasing the concentration of the more floatable particles. This is the basis of "rewashing" in flotation practice.

Flotation is essentially a differential process, and, apart from the general separation of mineral from gangue, different types of mineral may be separated from each other, as well as certain types of gangue from other types. Although gangues are of a relatively low order of floatability, they exhibit differences among themselves, since they may consist of quartz, secondary silica, quartzites, felspars, barytes, and numerous silicates, all with different surface properties and

contact angles. Thus mica may be separated from felspar, barytes from quartzite, and oxides and carbonates from silicates.

Differential flotation may be achieved by the use of selective chemical methods to alter the surface properties of one or more constituents of an ore, as in the Horwood process. On roasting, galena readily undergoes surface oxidation to become coated with a layer of lead sulphate, and so becomes less floatable, but zinc blende may be unaffected by the same treatment. Separation of blende from galena is, therefore, facilitated. Separation is, however, usually incomplete, owing to the difficulty, in practice, of achieving complete surface oxidation of quantities of very small particles. This is particularly the case when, as is usual, the particles are in a flocculated condition. The employment of soluble chromates to coat galena with a film of lead chromate is of theoretical rather than of practical importance. Other chemical modifications employed are the use of fatty acid emulsions for combination with certain metallic oxides and carbonates, which then become filmed with metallic soaps, and the use of soluble sulphides to produce sulphide films on oxidised metals. Both processes, however, suffer from the disadvantage of having to deal with finely-divided particles of the ore pulp, and from the liability of the films to be stripped off by the agitation necessary in all froth-flotation practice.

The physical methods of governing differential flotation have already been considered in detail. They may be summarised as follows: (1) Accentuation of flocculation and deflocculation; (2) limitation of film stability; (3) modification of surface properties (as oiling); (4) control of bubble surface by "overcrowding"; (5) variation of particle size. With regard to the latter factor, Sulman quotes an interesting experience with an Australian blende-galena product. With an elutriated product (equal-falling particles, in which the blende particles were about twice as heavy in water as the galena particles) the galena floated first with an unstable froth, and most of the blende was recovered in a second, stabilised, froth. On the other hand, when a screened product was used (in which the blende particles were only half as heavy as the galena particles) the blende particles, being lighter, were recovered first in an unstable froth.

CHAPTER XXII

FROTH FLOTATION : PROCESSES

GENERAL

THE reagents which are suitable for sulphide mineral flotation are also generally suitable for coal flotation, since both coal and sulphide ores have comparatively non-oxygenated surfaces, and are more readily wetted by oil than by water. The dirt usually associated with coal behaves similarly to the gangue of ores in being more readily wetted by water than by oil. The addition of a frothing agent such as cresol, aided by mechanical agitation, produces a bubble system in the body of the liquid. A coal particle in contact with an air bubble is, in effect, equivalent to a solid particle at a liquid/air interface, and the angle of contact of the air bubble at the liquid/solid interface will be large. The bubble will, therefore, be stably attached to the solid particle (conditions similar to those of Figs. 176 and 177). If the particle is not too heavy, the buoyancy of the bubble will carry it to the top of the liquid, and the coal particles will therefore be concentrated in the froth. With shale, the contact angle at the solid/liquid interface will be small, and a bubble would be unstable and easily dislodged from a solid particle when, as in practice, the mass is violently agitated. Shale particles are thus not borne to the surface by attached bubbles, and separation of coal and shale is possible.

Pyrites requires special consideration in coal flotation. It forms a large contact angle with water and so is floatable, but since flotation is essentially differential, it is possible to ensure that coal is floated in preference to pyrites by taking advantage of principles previously considered, namely, those relating to overcrowding and size. The application of the principle of overcrowding needs no further amplification, but the size factor may be briefly reconsidered.

If equal-sized particles are used, there would be little difficulty in preferentially floating coal from the pyrites, which is many times heavier than coal. If the sizing were not very close, however, small pyritic particles might be floated with the coal. Indeed, it seems not unlikely that the difficulty in separating those particles which just cannot be separated by ordinary methods of classification, namely, "equal-falling" particles, might be greater in froth flotation than is the case when methods dependent essentially upon density differences are employed.

Differential flotation may be applied to coal by the choice of reagents. It has been shown, for example (Chapman, *Fuel*, 1922,

1, 52), that, when using kerosene, the resultant froth contained 76 per cent. of bright coal (clarain and vitrain) and only 24 per cent. of dull coal (durain). With phenol, the froth contained only 20 per cent. of bright coal and as much as 80 per cent. of dull coal, in whatever order the reagents were added.

In experimental work on the froth flotation of coal, cresol (cresylic acid) is the usual frothing agent used, and paraffin oil is used for froth stabilising. In practice, suitable agents are cresol and gas oil, or one of the higher boiling fractions of distilled tar, containing both cresols and neutral oils. The effluent water from the naphthalene-scrubbing tower used in the Otto direct ammonia-recovery process on coke-oven plants is found to contain sufficient tar acids (frothing agents) and neutral oils (froth stabilisers) to be used for froth-flotation practice, without recourse to separate oil additions. It is said that effluent ammonia liquors may also be used, and since the ammonia liquor itself has been in contact with the tars produced during carbonisation, it is to be expected that, even after distillation, ammonia liquor will contain tar acids and neutral oils, but the excessive content of inorganic salts would probably make its use undesirable.

In coal-cleaning practice, the particular *rôle* which froth-flotation processes are best able to fill is in the cleaning of small coal, which is only imperfectly cleaned by many other processes of coal cleaning. Indeed, the upper limit of size for successful flotation is $\frac{1}{10}$ in., and coal exceeding this size must either be crushed before washing, or else be washed by other means. In the early days of its development, when flotation practice had achieved some spectacular triumphs in ore-dressing practice, the possibilities of its application to coal cleaning appealed to the imaginations of many whose predictions owed more to the fertility of this source of inspiration than to the established facts of large-scale operation. It was proposed that coking coal, which is usually crushed after washing for charging to the ovens, should be crushed before washing and cleaned by a froth-flotation process. The practical disadvantages of such a procedure are that the large pieces of free dirt and pyrites would have to be crushed and, because of their relative hardness, would make the crushing operation more expensive ; the coal which was not crushed through $\frac{1}{10}$ in. mesh would have to be recovered from the flotation residues by other washing methods or else the residue would have to be recrushed. The problem of handling large quantities of small coal throughout the washery operations and the final dewatering present great technical difficulties. The cost of washing all coking coal in quantities of, say, 100 tons per hour by a froth-flotation process, with the large number of units, the considerable amount of attention required, and the special knowledge demanded, apart from the cost of reagents and the royalty payable, would make the commercial process compare most unfavourably with other and simpler processes for coal-cleaning.

The claim that froth-flotation treatment produces cleaner coal than that produced by other cleaning processes has not always been substantiated in practice. Such a claim is often based on the results of an experimental laboratory froth-flotation unit, which are compared with the results of cleaning coal in jig washers dealing with 100 tons of coal per hour. It is manifestly absurd to compare results obtained under such different conditions, and the only comparison worth considering is on a basis of similar throughput. If coke of particularly low ash content is required, suitably-cleaned coal can be prepared by most methods of washing when dealing with all coal up to, say, $\frac{5}{8}$ in. delivered to the washery.

A rational method of procedure would be to clean all the raw coal by some efficient type of "gravity" washer and to recover only the coal floating in a liquid of S.G. 1·4. The middlings and dirt might then be crushed to liberate some of the intergrown coal, and the crushed material might be cleaned by froth flotation, or other process which deals efficiently with fine materials. In this way a cleaned product suitable for admixing with the coal cleaned by other means could be produced, and a particularly high grade of clean coal would be available.

Alternatively, to avoid the cost and difficulty of crushing the dirt and middlings, it might be arranged that the slurry produced by ordinary wet-washing processes could be treated by a froth-flotation process (or some other efficient process for cleaning very fine coals) and admixed with the rest of the clean coal. This method would not yield such a clean product as the first one, but, when the middlings fraction is only small, it is an easy one to adopt. It is used at Clifton, Lancs., to clean the slurry produced in a Baum washery. By adopting this scheme, clean coals of 4 to 5 per cent. ash content may be obtained.

Results

Some of the results obtained by froth-flotation cleaning are recorded in the following tables. Bury, Broadbridge and Hutchinson (*Trans.* Inst. Min. Eng., 1920, 60, 243) record results (reproduced in Table 107) when using an experimental plant in Durham.

All the materials treated were previously crushed so as to pass through $\frac{1}{10}$ in. screen. The results obtained with the coking coals are very good; those for the non-coking coals, slacks and slurries are not so good. Non-coking coals (excluding semi-anthracite and anthracite) have a higher oxygen content than coking coals, and are therefore more easily wetted by water and are not so readily separated from shale. The results obtained for the washery wastes show that appreciable quantities of combustible matter may occasionally be recovered by crushing and rewashing, although the quality of the product is inferior, so that it could not be used alone for processes requiring high-grade materials. It might with advantage, however, be mixed with the main clean coal fraction for many

TABLE 107.—RESULTS OF CLEANING ENGLISH COALS BY FROTH FLOTATION

Description.	Raw Coal. Ash per cent.	Clean Coal.		Refuse.	
		Weight per cent.	Ash per cent.	Weight per cent.	Ash per cent.
Coking, No. 1 . .	12·4	87·8	3·8	12·2	72·4
,, No. 1 . .	24·2	75·9	5·2	24·1	78·5
,, No. 3 . .	15·8	83·2	5·4	16·8	76·0
Non-coking, No. 1 .	25·5	73·8	8·9	26·2	84·5
,, No. 2 .	27·0	68·3	9·0	31·6	76·0
,, No. 3 .	21·8	69·2	7·3	30·1	74·4
,, No. 4 .	28·2	66·7	9·4	27·5	78·8
Slack, No. 1 . .	30·0	68·9	12·2	31·1	80·0
,, No. 2 . .	30·5	71·0	9·6	29·0	86·5
Slurry, No. 1 . .	35·5	60·2	8·1	39·8	74·0
,, No. 2 . .	33·8	48·5	11·7	51·5	78·0
,, No. 3 . .	21·5	83·8	9·9	16·2	81·5
,, No. 4 . .	45·2	59·0	12·5	40·5	82·8
Washery waste, No. 1	74·0	16·8	13·0	83·2	86·6
,, ,, No. 2	40·3	53·0	7·9	47·0	75·8
,, ,, No. 3	61·2	30·2	10·1	37·8	86·1
,, ,, No. 4	76·0	14·5	13·5	83·0	87·6
,, ,, No. 5	62·2	24·3	9·6	70·7	84·8
,, ,, No. 6	75·0	16·2	14·0	83·8	88·5
,, ,, No. 7	63·8	28·7	10·8	71·3	86·5

TABLE 108.—RESULTS OF CLEANING DERBYSHIRE COALS BY FROTH FLOTATION

Description.	Raw Coal.		Washed Coal.		
	Ash per cent.	Sulphur per cent.	Weight per cent.	Ash per cent.	Sulphur per cent.
Slack, No. 1 . .	19·4	2·47	76·2	5·5	2·38
,, No. 2 . .	20·4	2·97	—	4·2	2·21
,, No. 3 . .	20·5	2·81	70·0	4·4	1·93
,, No. 4* . .	21·2	2·32	60·4	7·6	1·93
Slurry, No. 1 . .	21·1	2·84	67·0	7·0	2·44
,, No. 2 . .	23·2	—	70·8	7·1	—
,, No. 3 . .	17·5	2·51	—	10·0	2·29

* Raw coal contained 67·8 per cent. through $\frac{1}{16}$ in. mesh; all others were ground so as to pass through $\frac{1}{10}$ in. mesh.

purposes. The quantity of material recovered from the refuse is small in comparison with the bulk of the cleaned coal, and the resulting increase in the ash content would often be compensated by the higher gross yield.

Results obtained for Derbyshire coals by Lee and Whitehead (*Gas World*, 1924, June 7) on a laboratory scale are recorded in Table 108.

Table 109 records the results of treating a number of Spanish coals (Louis, *Colliery Engineering*, 1924, p. 427).

TABLE 109.—RESULTS OF CLEANING SPANISH COALS BY FROTH FLOTATION

Description.	Raw Coal.	Washed Coal.		Refuse.
	Ash per cent.	Weight per cent.	Ash per cent.	Ash per cent.
Penarubia fines . .	30	66	10	70
Mariana fines . .	32	50	11–12	55
Ujo slurry . . .	32	65	9	75
Turon slurry . .	32	70	9	75
Modesta wash water .	30–32	—	10–12	55

In Table 110 results obtained by Dessagne for French coals are recorded (*La Technique Moderne*, 1923, 15, 554).

TABLE 110.—RESULTS OF CLEANING FRENCH COALS BY FROTH FLOTATION

Description.	Raw Coal. Ash per cent.	Washed Coal. Ash per cent.	Refuse. Ash per cent.	Reagents Used.
Slurry No. 1 .	36·5	10·9	71	Turpentine, 2½lb./ton.
„ No. 2* .	33·0	11·8	67	Crude creosote, 4¼ lb./ton.
„ No. 3 .	35·0	11·1	68	Turpentine + 10 p.c. benzol, 3 lb./ton.
„ No. 4 .	38·0	12·0	65	—
Coal No. 1 .	29·0	11·0	77	Turpentine, 2¼lb/ton.

* The percentage of ash in the products from successive boxes was 4, 6·5, 9, 12, 17 and 21.

Dessagne records an attempt to obtain a saleable product (with ash less than 12 per cent.) from washery waste containing 35 to 45 per cent. ash. Preliminary experiments, varying the type of oil

used, only resulted in the production of coal containing from 15 to 26 per cent. of ash. The poor results were attributed to the presence of middlings and of disintegrated clay, which easily floated of its own accord without being brought to the surface in bubbles. An attempt was made to remove the disintegrated clay by elutriation, but the ash content of the raw material was only decreased from 43 to 38 per cent. Fine grinding was resorted to in order to overcome the large contribution of ash from the middlings, and the following results were obtained, using turpentine as the reagent :—

Size (mm.).	Cleaned Coal, Ash per cent.	Refuse. Ash per cent.
< 2 . . .	17	62
< 1 . . .	15	64
$< \frac{1}{2}$ . . .	19	58
$< \frac{1}{5}$ . . .	20	55

Rewashing the coal was also practised without success, and failure of the process was, therefore, admitted.

L. Bloum-Picard (*La Revue Industrielle*, 1926, May, June, July) describes an attempt to recover fine coal from a French pit-heap of washery waste containing some 3,000,000 tons of material with 60 to 65 per cent. of ash. A froth-flotation plant of 1,000 tons per day capacity was used and the raw material was sieved to remove the large pieces of dirt and crushed to less than 2 mm. ($\frac{1}{12}$ in.) size by hammer mills. The percentage recovery of fine coal was 20 to 22 per cent., with an ash content of 18 per cent., the refuse having 72 to 74 per cent. of ash. The coal was too much oxidised to use for coking purposes, and briquettes were made from it with the aid of oil (by the Trent process).

O. Schäfer (*Stahl u. Eisen*, 1925, 45, 49) records the results of cleaning German slurries by froth flotation. His results are given in Table 111.

The plants treating the first four slurries were Minerals Separation plants, each of eight or ten cells. The remaining slurry was treated in an Ekof plant. Schäfer drew attention to the enhanced yields of coal with a low percentage of ash, obtained by using froth flotation in addition to the ordinary "gravity" washing process. The fine coal of Waldenburg, when washed in a "gravity" washer, would give a yield of only 56 per cent. of material with an ash content of 5·5 per cent. ; by cleaning the 10 to $\frac{1}{2}$ mm. ($\frac{3}{8}$ to $\frac{1}{50}$ in.) coal by a gravity washer, 53 per cent. of coal with 5·1 per cent. of ash would be obtained, and, by treating the $\frac{1}{2}$ mm. to 0 ($\frac{1}{50}$ in. to 0) size by froth flotation, 15·2 per cent. of coal with 7·0 per cent. of ash would also be obtained, giving a gross yield of 68·2 per cent. of coal of

TABLE 111.—RESULTS OF CLEANING GERMAN SLURRIES BY FROTH FLOTATION

Description.	Raw Slurry. Ash per cent.	Washed Coal. Weight per cent.	Washed Coal. Ash per cent.	Refuse. Ash per cent.	Capacity. Tons/hour.
Zwickau, Saxony . .	35–40	50–55	8–9	20–25	12
Mölke, Silesia . .	20–23	45–50	8–9	30–35	35
Gelsenkirchen, Westphalia	9–10	92–94	5–6	75–80	15
Altenwald, Saar . .	22–25	70–72	7–8	65–75	10
Mont Cenis, Westphalia .	25–30	50	7–5	75	5

5·5 per cent. ash content. When washing the same coal to 6·5 per cent. of ash, an increased yield of 15·7 per cent. would be obtained if the finest coal were cleaned by froth flotation. Similar results are recorded for a Westphalian "gas-flame" coal.

W. Randall (*Rec.* Geol. Surv. India, 1924, 56, 225) records the results of cleaning Indian coals by a froth-flotation process on a laboratory scale. Indian coals, in general, contain mineral matter which is very intimately associated with the true coal material, and there is an insensible transition from the coal of lowest ash content to the coal of highest ash content, the specific gravity varying in a like manner. Separation in "gravity" washers is, therefore, diffi-

TABLE 112.—RESULTS OF CLEANING INDIAN COALS BY FROTH FLOTATION

Description.	Raw Coal. Ash per cent.	Washed Coal. Weight per cent.	Washed Coal. Ash per cent.	Refuse. Weight per cent.	Refuse. Ash per cent.
Jharria, No. 16 seam (coking) face sample, run-of-mine .	22·4	60	11·8	40	38·3
Do. through 1 in. fines from crushed run-of-mine coal .	20·1	60	10·2	40	34·9
Do. through ½ in. fines from crushed run-of-mine coal .	19·2	60	9·5	40	33·8
Bokaro, Kargali seam (coking) face sample, run-of-mine .	22·3	50	14·4	50	30·2
Do. through ½ in. fines from crushed run-of-mine coal .	20·6	50	9·6	50	31·6
Barakar, Laikdih sm. (coking) face sample, run-of-mine .	14·0	60	10·4	40	19·4

cult, and, in any event, crushing to liberate some of the intergrown dirt is advisable. The fine crushed material then obtained cannot be very successfully treated in gravity washers. For the experiments made, the coal was crushed so as to pass through a $\frac{1}{20}$ in. screen. Typical results are recorded in Table 112.

The fractures and cleavage planes of the seams occurred in bands of vitrain and clarain which had a much lower ash content than the harder durain. The slack formed in mining, therefore, had a lower ash content than the run-of-mine coal, and it was shown that improved qualities of slack for coking could be prepared by crushing the run-of-mine coal and using the slack less than 1 in. or $\frac{1}{2}$ in.

A comprehensive set of results for cleaning coal by froth flotation on a large scale was recorded by Scoular and Dunglinson (*Trans. N. of Eng. Inst. of Min. Eng.*, 1924) for Cumberland coals. One of the seams treated had the composition recorded in Table 113 when crushed through 10 mesh (about $\frac{1}{13}$ in. for the sieves used). For the float and sink tests, the coal of 10–20 mesh size only was used.

TABLE 113.—COMPOSITION OF RAW COAL

S.G.	Weight per cent.	Ash per cent.	Sulphur per cent.
< 1·25	5·2	1·15	1·02
1·25–1·30	54·0	1·91	1·09
1·30–1·35	10·7	6·14	1·71
1·35–1·40	5·0	9·83	1·85
1·40–1·45	2·5	14·12	1·93
1·45–1·50	1·4	19·08	1·99
1·50–1·55	0·9	24·79	2·05
1·55–1·60	0·6	28·84	2·01
1·60–1·80	1·4	40·23	2·60
1·80–2·40	2·7	64·71	2·86
2·40–2·90	9·8	78·28	2·02
> 2·90	3·8	71·52	20·55

The screen analysis of the coal treated was as follows :—

Size of Mesh.	Per cent. by Weight.
10–20	13·1
20–60	40·9
60–100	14·3
100–200	16·2
< 200	15·5

The coal was treated in an eight-cell apparatus (Minerals Separation) with two additional mixing boxes. The ground space

occupied by the apparatus was 37 ft. by 15 ft. Cresylic acid was used as a frothing agent in an amount of 0·77 lb. per ton of coal; 0·37 lb. of gas oil per ton of coal was also used as a froth stabilising agent. The composition of the products in the froth from the first seven boxes is recorded in Table 114.

TABLE 114.—COMPOSITION OF PRODUCTS FROM DIFFERENT CELLS OF A FROTH-FLOTATION PLANT

Cell. No.	Concentrates.		S.G. < 1·6.*			S.G. > 1·6.*		
	Ash per cent.	Sulphur per cent.	Weight per cent.	Ash per cent.	Sulphur per cent.	Weight per cent.	Ash per cent.	Sulphur per cent.
1	3·02	1·27	98·6	2·33	1·05	1·4	51·2	9·27
2	3·16	1·35	98·5	2·38	1·20	1·5	54·5	11·30
3	3·26	1·30	98·2	2·33	1·11	1·8	54·3	11·67
4	3·73	1·37	97·9	2·64	1·15	2·1	55·1	11·40
5	4·73	1·59	97·0	3·19	1·25	3·0	54·7	12·72
6	6·02	1·78	95·4	3·52	1·31	4·6	57·8	11·54
7	7·15	2·12	94·4	4·00	1·25	5·6	60·2	16·75

* On 10-20 mesh coal.

The total cleaned coal contained 5·30 per cent. of ash. The size of the various fractions is recorded in Table 115.

TABLE 115.—SIZE ANALYSES OF THE PRODUCTS FROM THE DIFFERENT CELLS OF A FROTH-FLOTATION PLANT

Screen Mesh.	Cell No.						
	1.	2.	3.	4.	5.	6.	7.
> 10 . .	—	—	—	0·7	3·1	8·9	13·6
10–20 . .	3·0	3·9	12·2	20·7	42·4	51·7	55·4
20–60 . .	33·7	37·6	48·0	46·0	35·5	27 0	20·9
60–100 .	18·7	17·0	13·9	10·6	5·2	3·1	3·2
< 100	44·6	41·5	25·9	22·0	13·8	9·3	6·9

It will be observed that the sulphur content of the material of S.G. >1·6 in all the fractions is high, showing that the small amount of dirt remaining in each fraction includes a large proportion of pyrites. Another point of interest is the gradually increasing size of the coal floated in successive cells, showing that the influence of gravity is not unimportant, even in a froth-flotation process.

The costs of crushing the coal so as to pass through $\frac{1}{10}$ in. screen were found to be much higher than was anticipated, since the bars of the Carr crusher wore out more rapidly when crushing dry coal before washing than in the customary procedure of crushing after washing. This increased cost was obviated by crushing the coal less drastically and then screening over a $\frac{1}{4}$ in. screen, the undersize being then treated in the froth-flotation plant. In this washer, only the coal up to 10 mesh size was recovered, the 10 mesh to $\frac{1}{4}$ in. size coal passing away with the dirt, which was then screened over a 20 mesh screen and the oversize cleaned separately on a Plat-O concentrating table to recover the coarse coal. In this way the crushing and screening costs were reduced to the same level as when crushing was carried out after washing in the normal coke-oven practice.

Further interesting results of the use of froth flotation for coal cleaning are recorded by W. Guider (*Journ.* Soc. Chem. Ind., 1927, 46, 238). His results were obtained with the concentrated slurry from the conical settling tank of a Baum washer. An analysis of the coal used is recorded in Table 116.

TABLE 116.—ANALYSIS OF RAW COAL FED TO BAUM WASHERY

Size (in.).	Per cent. by weight.	"Pure" Coal < 1·35 S.G.	Middlings 1·35–1·60 S.G.	Dirt > 1·60 S.G.
$2\frac{1}{2}$–1	28·0	20·9	2·5	4·6
1–$\frac{1}{2}$	19·0	14·1	1·9	3·0
$\frac{1}{2}$–$\frac{1}{4}$	25·0	17·1	3·2	4·7
$\frac{1}{4}$–$\frac{1}{10}$	15·0	10·6	1·5	2·9
$\frac{1}{10}$–$\frac{1}{20}$	3·6	2·2	0·3	1·1
< $\frac{1}{20}$	9·4	5·1	1·7	2·6
Total	100·0	70·0	11·1	18·9

From these results it might be considered that the coal was not more difficult to wash than most coals, and this is borne out by the fact that the fine washed coal through $\frac{1}{2}$ in.* contained about 6·7 per cent. of ash before the slurry was added to it. The results were, however, entirely modified by the characteristics of the shale associated with the coal and which readily disintegrated in water and tended to form a clay slip. Thus the quantity of very fine dirt which was formed in the wash water was much more than

* The fine coal was screened from the coal before washing and treated directly in one wash-box. This practice did not follow the orthodox Baum method of first passing it through the primary box and then rewashing it.

would be expected from the preliminary float and sink analysis. For example, it was found in practice that, two days after commencing with clean water, the water leaving the cleaned product during dewatering contained 16 per cent. of solids with an ash content of about 30 per cent. The slurry from the settling tank after dewatering contained 14·4 per cent. of ash, and was sufficient in quantity to raise the ash content of the washed coal to 10 per cent. Moreover, the sludge which settled to the bottom of the settling tank overnight, and which was withdrawn every morning, contained about 30 per cent. of ash. It is interesting to note that, of this sludge, about 20 per cent. by weight was less than $\frac{1}{120}$ in. size, and contained about 65 per cent. of ash, and appears to have consisted entirely of disintegrated shale. In addition to the contamination of the washed coal, the specific gravity of the washery water rose to 1·1, and reduced the efficiency of washing.

Referring to Table 116, it will be observed that the raw coal contained 3·7 per cent. of material less than $\frac{1}{10}$ in. size which sank in a liquid of S.G. 1·60, and probably had an ash content of about 69 per cent. With a feed of 80 tons of coal per hour, it would be necessary to remove from the washer the 3 tons of this fine dirt which entered with the raw coal per hour, in addition to the considerable quantities resulting from the disintegration of the shale when in contact with water. To run this material to waste from the slurry dewatering screen was considered to be undesirable, owing to the quantity of coal which would also be lost (about 5 tons per hour). A Minerals Separation plant was therefore installed to treat the slurry from the settling tank.

The plant comprised six cells with a capacity of 25 tons per hour. The feed was raised from the base of the settling tank to the flotation unit by an injector using fresh water. The feed contained 25·6 per cent. of solids in suspension, with the following composition (Table 117) :—

TABLE 117.—COMPOSITION OF FEED TO FROTH-FLOTATION PLANT

Size (in.).	Per cent. by weight.	< 1·35 S.G.	1·35–1·60 S.G.	> 1·60 S.G.
> $\frac{1}{4}$	1·8	1·6	0·2	—
$\frac{1}{4}$–$\frac{1}{10}$	8·2	6·6	1·2	0·4
$\frac{1}{10}$–$\frac{1}{20}$	8·7	6·7	1·3	0·7
< $\frac{1}{20}$	81·3	46·3	18·7	16·3
Total	100·0	61·2	21·2	17·4

The material over $\frac{1}{10}$ in. size (10 per cent. of the feed), which could not be floated in the Minerals Separation plant, appeared in the refuse, and was removed by passing it over a wedge-wire screen, the oversize being added to the fines wash-box of the Baum washer. The undersize passed to waste and had the following composition (Table 118) :—

TABLE 118.—COMPOSITION OF REFUSE FROM FROTH-FLOTATION PLANT

Size (in.).	Per cent. by weight.	< 1·35 S.G.	1·35–1·60 S.G.	> 1·60 S.G.
$> \frac{1}{10}$	1·6	1·0	0·3	0·3
$\frac{1}{10}-\frac{1}{20}$	4·2	1·4	1·4	1·4
$< \frac{1}{20}$	94·2	3·4	7·9	82·9
Total	100·0	5·8	9·6	84·6

The refuse therefore contained nearly 6 per cent of "pure" coal and 10 per cent. of middlings, but it should be remembered that this is calculated on the reject from the froth-flotation plant. When calculated on the feed to the flotation plant the loss is approximately 1 per cent., and only 0·1 per cent. of the total feed to the washery. The composition of the concentrate is recorded in Table 119.

TABLE 119.—COMPOSITION OF RECOVERED COAL FROM FROTH-FLOTATION PLANT

Size (in.)	Per cent. by weight.	< 1·35 S.G.	1·35–1·60 S.G.	> 1·60 S.G.
$> \frac{1}{20}$	5·0	4·5	0·5	—
$\frac{1}{20}-\frac{1}{60}$	38·0	33·5	4·0	0·5
$< \frac{1}{60}$	57·0	37·0	15·0	5·0
Total	100·0	75·0	19·8	5·5

The ash content of the concentrate was 6 per cent. and its water content 50 per cent. The dirt remaining amounted to 5·5 per cent., but this was of very small size, and may have come from the dirty water with which the coal was contaminated.

Cresylic acid was at first used in quantities of 1·04 lb. per ton

of recovered coal with 1·03 lb. of creosote oil in addition, the cost of the reagents being 3·2*d*. per ton of recovered coal. Later, the use of cresylic acid was abandoned, and the amount of creosote oil was increased to 2·7 lb., with 0·3 lb. of gas oil, and, finally, 1·7 lb. of creosote oil only was used at a cost of 1·5*d*. The total costs were given as being (per ton of recovered coal) :—

	Pence
Interest on capital (5 per cent.) . .	5·60
Depreciation (7 per cent.) . . .	7·84
Royalty	4·00
Power.	3·84
Wages (one man)	1·85
Reagents	1·50
Total	24·7

The increased cost per ton of slack (through ½ in.) was stated to be 6·1*d*., but this cost is higher than has been experienced elsewhere.

The benefits of applying froth flotation were stated to be that a washed slack of 6 per cent. ash content was regularly obtained, and that the larger sizes were not contaminated by dirty washing water. The sludge was no longer withdrawn periodically from the settling tank after standing overnight, and an additional 30 tons of coal per week were recovered. The disadvantage was that the washed coal retained 18 per cent. of water, and it was found to be impossible to dewater the froth-flotation concentrate on an ordinary wedge-wire screen.

In considering these results, it should be borne in mind that fine coal, as treated in froth-flotation practice, contains fewer interstratified particles than the larger sizes of coal treated by other processes. One would therefore expect to find that froth flotation could give a greater operating efficiency than other processes, but this scarcely appears to be the case, probably on account of mechanical entanglement of dirt in the washed coal due to the large number of particles treated, or because of the presence of fine suspended dirt particles in the water that remains admixed with the concentrate in equal proportions when it leaves the flotation plant. Nevertheless, froth-flotation methods are a valuable addition to those which can be used for the cleaning of coal, in particular for the finer sizes of coal which are inevitably produced in mining operations.

PROCESSES

A number of designs of froth-flotation plant have been proposed, but it will be sufficient only to describe the principal ones.

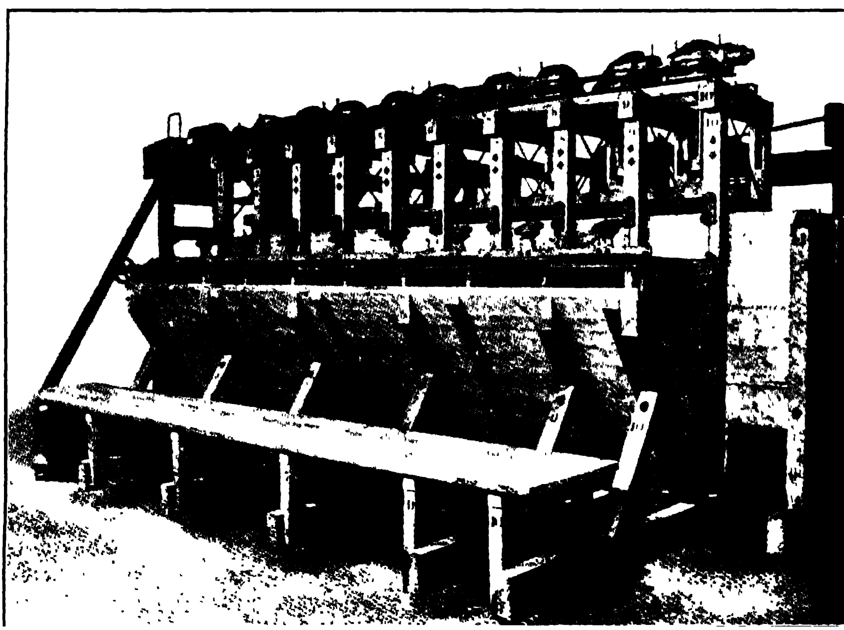

FIG. 180.—View of Minerals Separation Froth-Flotation Plant.

[To face page 129.

The Minerals Separation Plant.—A ten-box froth-flotation plant of Minerals Separation Ltd. is illustrated in Fig. 180. Each box consists of a mixing cell (in which a paddle causes the necessary agitation for mixing the feed with the reagents, and the entrainment of air) and a froth cell (in which the necessary quiescent conditions are produced for the froth to rise to the surface of the cell, and overflow). A section through two boxes and a cross-section through an agitation and a froth cell, are shown in Fig. 181. The coal, crushed to a size less than $\frac{1}{10}$ in., is mixed with four to five parts of water, and is admitted to the agitation cell of the first box, where the reagents are added. The agitated and aerated mixture then passes to the corresponding froth cell, where the first froth is collected. The residue passes from the bottom of the first froth cell, through a circulating pipe to the agitation cell of the second

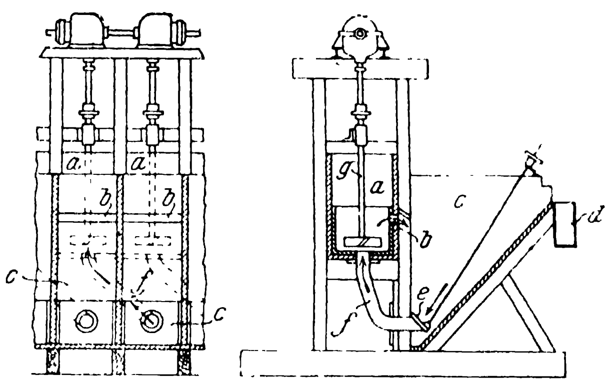

FIG. 181.—Minerals Separation Plant: Longitudinal and Cross-section.

box, and the residue from the second cell is similarly transferred to the third cell, and so on until the last cell is reached. The apparatus is usually constructed chiefly of wood. The paddles of the different cells are keyed to vertical spindles driven through bevel gearing by a shaft mounted rigidly on a strong framework above the agitation cells. At the discharge end of the circulating pipe, a valve, controlled by a handwheel, is fitted to regulate the rate of flow through the plant.

The frothing agent first added may be cresol in amounts varying from $\frac{1}{2}$ to 5 lb. per ton of coal, and, with an unstabilised froth, the cleanest coal may be recovered. By the addition, at a suitable later stage, of a stabilising agent (neutral oil) in amounts similar to those used for the frothing agent, the middlings may be recovered in a stabilised froth.

The cleaned coal contains up to 50 per cent. of water, and in one instance even on drainage for a fortnight the water content was only reduced to about 35 per cent. (Hanson, *Proc.* Cleveland Inst. Eng., 1922, p. 26). The drainage of the washed coal has proved to be one

of the most serious stumbling-blocks in the use of the froth-flotation process for coal, and is discussed separately in a later chapter. It may be said, however, that ordinary dewatering or drainage methods have not proved adequate. Centrifugal separators are not much more satisfactory, and the use of filters (*e.g.* the Oliver) is expensive and not wholly satisfactory. It would appear that the only reliable methods are to remove the water by heat or to make use of the preferential wetting of coal by oil by displacing the water from the particle surfaces, followed by a pressure or suction treatment to remove the loosely-entangled water from the coal mass.

The capacity of the Minerals Separation plants is only small. The smallest plant has a capacity of 5 tons per hour and the largest 25 to 40 tons per hour. The power requirements for a 25 tons per hour plant are about 50 h.p., or 2 h.p. per ton. Thirty-six Minerals Separation plants have been installed for coal cleaning with capacities varying from 5 to 30 tons per hour. Of these plants, 14 have been erected in Spain, 12 in Germany, 2 in Belgium, 1 in France, and 7 in Great Britain. Those in Great Britain are situated at Teams and Randolph, Durham; Bargoed and Aberaman, South Wales; Low Laithes, Yorkshire; Clifton, Lancashire; and Carron, Scotland.

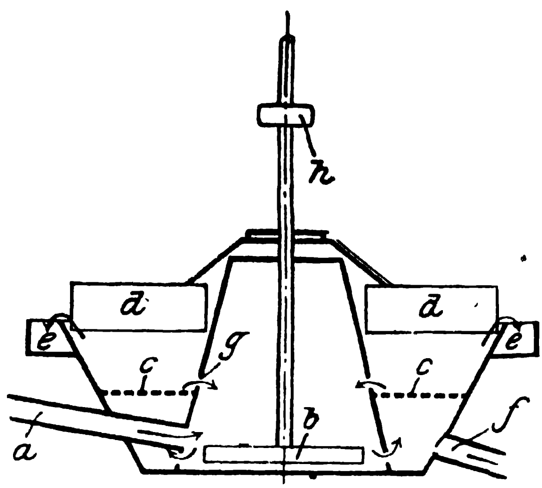

FIG. 182.—Kleinbentinck Froth-Flotation Unit.

The Kleinbentinck Plant.—The Kleinbentinck apparatus is used at a number of the collieries of the Dutch State mines and elsewhere. Each unit, as shown in Fig. 182, includes a mixing and a frothing chamber, both made of metal. It consists of an outer chamber or pan in the shape of an inverted truncated cone 0·5 metre (1 ft. 7½ in.) high and 1·8 metres (5 ft 10¾ in.) diameter at the top. A second chamber (or bell) of truncated cone shape is centrally disposed in the outer chamber and contains a screw, *b*, driven by a pulley, *h*, at a speed of 500 revs. per min. The small coal (slurry) is admitted to the inner mixing chamber, or bell, through the pipe, *a*, and the oil is added at the top of the bell. The materials are thoroughly mixed in the bell, and, together with entrained air, pass through holes disposed close to the base of the bell into the outer frothing chamber.

At about half the height of the outer chamber an annular 6 mm. ($\frac{1}{4}$ in.) screen is fixed horizontally to ensure quiescent conditions in the space above it. The frothed material passes through the screen and is scraped off mechanically into an annular launder, *e*. The non-floating material returns through a series of 25 mm. (1 in.) openings, *g*, to the bell, where it is reagitated. Finally, the refuse is withdrawn through the exit, *f*. By controlling the admission of the feed, the material already in the apparatus may be circulated a number of times. Alternatively, the unfloated material may be re-treated in further units.

According to the inventor, Kleinbentinck, the chief engineer of the Dutch State mines, the power requirements are 2·7 h.p. per ton (Berthelot, *Bull.* Soc. d'Enc. pour l'Ind. Nat., 1925, 124, 48). The oil used is a mixture of 1 part of naphthalene oil and 2 parts of anthracene oil from tar distillation, to the extent of about 1$\frac{3}{4}$ lb. per ton of slurry. At the Emma colliery of the Dutch State mines, the

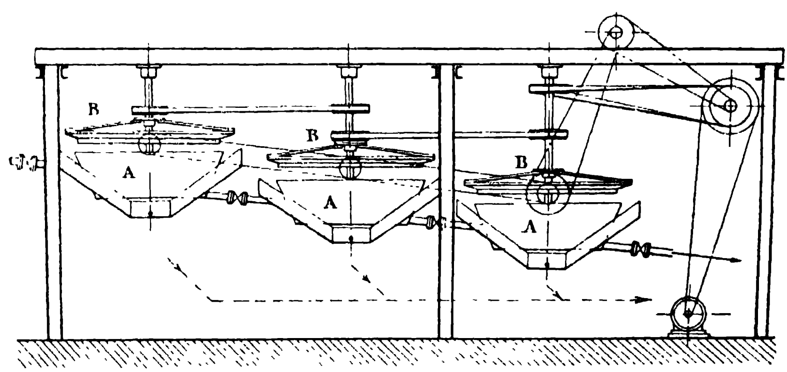

FIG. 183.—Arrangement of Kleinbentinck Units at Emma Colliery, Dutch State Mines.

raw slurry has an ash content of about 25 per cent. ; the froth from the first unit has an ash content of 8·5 per cent., and that from the second 7·0 per cent., the combined residues having 36 to 40 per cent. of ash. On rewashing this material, a product containing 10 to 12 per cent. of ash is obtained, leaving a residue with an ash content of 57 per cent. At the colliery named, the Kleinbentinck units are arranged as shown in Fig. 183. Six units are used to treat 12 to 15 tons of slurry per hour. The apparatus used to scrape off the froth is also illustrated. It consists of a crown, B, composed of four arms moving on rollers round the periphery of the outer pan, at a speed of 1·2 revs. per min.

At Aniche Colliery, France, there are two Kleinbentinck sets, each of three units, each set being capable of dealing with 15 tons of slurry of 0 to 1 mm. ($\frac{1}{25}$ in.) size, per hour. Another set of three units is used for rewashing (Sauvet, Rev. Ind. Min., 1926, 136, 355). At Aniche the slurry was successfully treated in three sets, the power requirements, for the gross throughput of 15 tons per hour, being

36 h.p. for the six washers and 24 h.p. for the rewashers, or 4 h.p. per ton-hour. A further 25 to 30 h.p. was used for the feed pump. The first three washers gave a floated product of 7·5 per cent. ash, the second three a product containing 11 per cent. ash, or a mixed product of 8·8 per cent. ash content. The rewashing apparatus is adjusted to give a floated product of 18 per cent. ash. The oil used is a tar oil (230 deg. to 270 deg. Cent. distillate) the amounts used being 18 lb. in the first three washers, and 6 to 9 lb. in the rewashing units, a total of 24 to 27 lb., or 1·6 to 1·8 lb. per ton of slurry.

The Ekof (Gröndal) Plant.—The German process of the Erz and Kohlen-Flotation (Ekof), working the patents of Gröndal, uses compressed air for agitation and aeration, in place of the mechanical means used in other processes. Fig. 184 illustrates an Ekof plant consisting of five cells. Compressed air is admitted from the pipe, *b*,

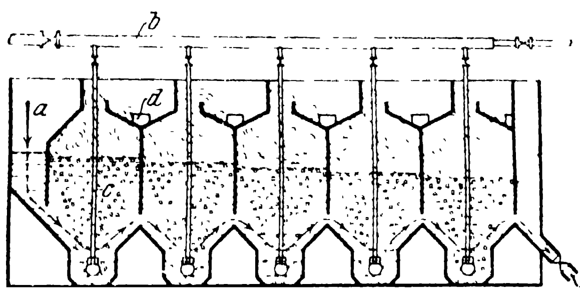

FIG. 184.—The Ekof Froth Flotation Plant.

to each unit through suitable valves, the air discharging into the bottom of each chamber, which is narrowed to produce a greater degree of agitation. The raw coal, previously mixed with the oil, is admitted at *a*, and passes through openings in the partitions between adjacent chambers, until the residue is finally discharged through the valve, *e*. The froth overflows from the top of each unit to collecting launders, *d*.

An Ekof plant is in operation at Mont Cenis Colliery, Westphalia, where, as previously noted, 5 tons of slurry containing 25 to 30 per cent. of ash are washed per hour, yielding 50 per cent. of a product with 7·5 per cent. of ash, 24 per cent. of middlings, with 14·5 per cent. of ash, and a residue of 26 per cent. containing 75 per cent. of ash. Twelve cells are used; the first six produce coal for coking, and the second six, the middlings fraction for boiler firing. The products are dewatered on jigging screens with a bed of fine coal. Air is supplied at a pressure of about 4 lb. per sq. in., and according to Berthelot (*loc. cit.*), 600 cu. m. (21,300 cu. ft.) are used per hour for a quantity of 5 tons of coal.

The Ekof plant was the first froth-flotation apparatus to be built

in Germany, being installed at Zwickau, Saxony, in 1923, by Fr. Gröppel, Bochum. O. Schäfer (*loc. cit.*) records that, with a through-

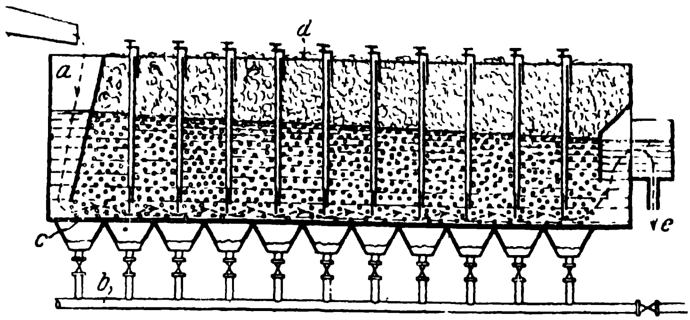

FIG. 185.—The Callow Froth-Flotation Plant.

put of 12 tons of raw slurry per hour, the power used was 50 h.p., or 4·2 h.p. per ton-hour.

The Callow Plant.—The Callow plant was first devised in 1914 by J. M. Callow, and was used by the Miami Company in America for the concentration of copper ores. The process is pneumatic, like the Ekof, the air being forced under pressure through the canvas bottom of the cells, to which the oiled pulp is admitted. A Callow plant is illustrated in Fig. 185.

The Humboldt Plant.—The Humboldt apparatus is illustrated in Fig. 186. Each unit consists of a mixing chamber and a froth chamber. Instead of using a paddle for agitation and circulation, a pump, *a*, is used, communicating both with the mixing chamber, through the pipe, *c*, and the frothing chamber, *e*. Compressed air is also supplied through the pipe, *b*, to the mixing chamber, where impingement on the plate, *d*, controlled by a handwheel, produces suitable agitation. The mixed material is forced into the frothing chamber from which the froth overflows at *f*. The unfloated material is drawn back into the mixing chamber for further agitation.

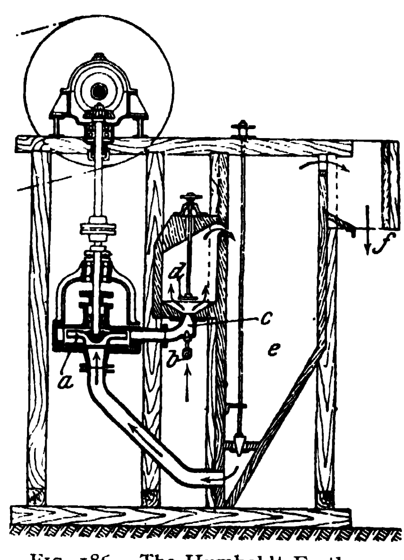

FIG. 186.—The Humboldt Froth-Flotation Unit.

The Coppée Plant.—The Coppée apparatus, illustrated in Fig. 187, consists of a mixing chamber, 2, and a frothing chamber, 7.

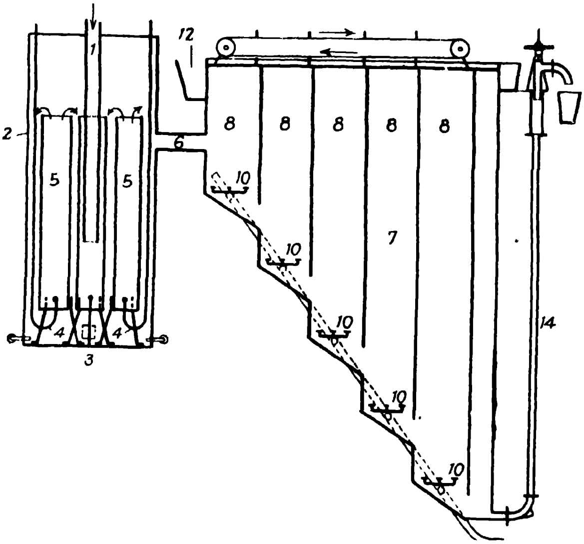

FIG. 187.—The Coppée Froth-Flotation Plant.

The coal, together with the water and oil, is admitted to the mixing chamber through the pipe, 1, and compressed air is forced in through a series of pipes inclined so as to produce suitable agitation. The well-mixed feed then enters the frothing chamber, 7, which is divided into a number of compartments, 8, by partitions which allow intercommunication at the bottom of the chamber. Compressed air is admitted to each compartment through the cocks, 10, and the froths are collected by projections from an endless belt into a launder, 12. The bottom of the frothing chamber is stepped to enable the residue from each compartment to pass to succeeding compartments, the final residue being discharged through the pipe, 14, fitted with a control valve.

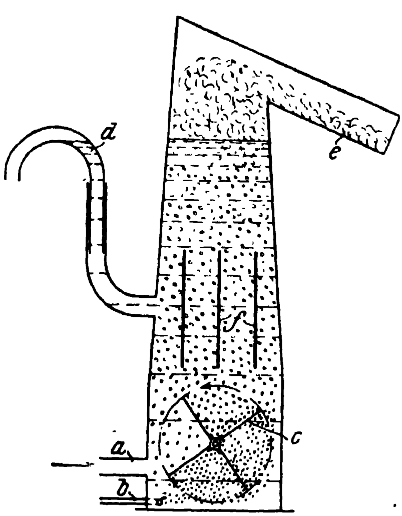

FIG. 188.—The Electro-Osmose Froth-Flotation Unit.

The Elektro-Osmose Plant.—The Elektro-Osmose apparatus,

illustrated in Fig. 188, consists of a simple cylindrical vessel to which the feed, admixed with water and oil, is admitted at *a*, and compressed air is blown in at *b*. A rotating mixer, *c*, entraps the coal and air and permits agitation before the discharge of the materials into the upper cylinder, where the plates *f* are fitted to ensure more quiescent conditions. The froth overflows at *e*, and the residue at *d*.

Thau (*Stahl und Eisen*, 1922, 42, 1155) records that, in the experimental plant of the Elektro-Osmose-Kohlenveredlung-Gesellschaft (Electro-Osmose Coal Purification Company), of Gelsenkirchen, Westphalia, over 100 samples of slurry from the Ruhr-Westphalian coalfield have been tested. He also records results, reproduced in Table 120, to show the progressive purification in successive units.

TABLE 120.—PURIFICATION OF SLURRY IN ELECTRO-OSMOSE PLANT

Description.	Raw Slurry. Ash per cent.	Ash per cent. in Products.					
		1st Unit.		2nd Unit.		3rd Unit.	
		Floatings.	Residue.	Floatings.	Residue.	Floatings.	Residue.
Alma . .	18·56	7·64	84·66	5·10	50·00	5·00	55·42
Möllerschächte	17·02	9·32	80·24	7·30	81·60	5·50	76·72
Mont Cenis .	28·05	14·72	84·24	10·36	67·00	7·09	48·82
Langenbrahn (anthracite)	12·78	4·20	53·64	3·20	38·62	1·60	68·26

With the Möllerschächte sample, the sulphur content of the slurry was reduced from 1·63 to 1·19 per cent., leaving 5·10 per cent. of sulphur in the refuse; with the Mont Cenis sample, the figures were 1·22, 1·10 and 2·25 per cent. respectively.

The Elmore Plant.—The Elmore vacuum-flotation apparatus was one of the earliest appliances used for the concentration of ores by flotation. It has not, so far as we know, been applied to the cleaning of coal on a commercial scale, but a number of coals have been tested in an experimental plant.

The results of two tests are recorded in Table 121.

In the latest plant the design has been altered from the simpler apparatus originally used for concentrating mineral ores by flotation, which is illustrated in Fig. 189. It consisted of a mixing chamber, *c*, to which the feed was admitted at *a*, and the oil at *b*. The paddle in the mixer revolved at a speed of 30 to 40 revs. per minute. The mixture was then drawn into the cone chamber, *e*, by means of

TABLE 121.—PURIFICATION OF FINE COAL IN EXPERIMENTAL VACUUM-FLOTATION PLANT

Description.	Ash per cent.		
	Raw Coal.	Clean Coal.	Refuse.
Scotch slurry .	15·4	4·2	58·5
Kent coal . .	8·8	3·8	56·0

a pump, and, at the reduced pressure in this vessel, air dissolved in the water was liberated in bubbles which carried the coal to the top of the cone, where it overflowed down a siphon tube, *g*. The refuse was siphoned off through the pipe *h*. The cone was 4 to 5 ft. diameter, and the feed pipe about 27 ft. long. In one plant 2·5 h.p. was used for the pump, cone-separator, and mixer.

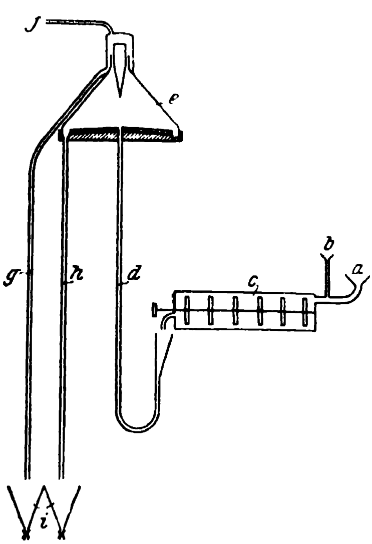

FIG. 189.—The Elmore Vacuum-Flotation Plant: Early Form.

The new form of apparatus used is illustrated in Fig. 190. Compared with the earlier plant, the cone separator has been modified, and a double cone, *a* and *b*, is now used. The upper cone may be regarded as the coal cone, and the lower one as the dirt cone. To the widest portion of this double cone, the coal, after admixture and agitation with oil, is now admitted from the hopper, *w*, through an expanding pipe, *u*. In the earlier apparatus (in which the feed pipe was quite narrow) it was found that the velocity in the feed pipe was too great. As the pulp rose in the feed pipe the pressure progressively decreased, thus causing liberation of the dissolved gases and their expansion in the pulp. The turbulent motion produced tended to detach air bubbles from the larger particles of coal which, therefore, sank and were rejected with the dirt. In the new plant a wide feed pipe is used and the expanding discharge compensates for the increasing volume of the air bubbles as the pressure is reduced. A fairly constant velocity is therefore maintained until the coal is discharged into the cone.

The coal particles are carried to the throat, *c*, of the upper cone and overflow into the launder, *d*, from which they are removed

through the siphon pipe, *g*, to the seal, *h*. The dirt particles, from which air bubbles are readily removed, sink in the lower cone. The feed pipe, *u*, is fitted with a number of concentric rings, *x*, between the spaces of which the heaviest dirt particles settle at once outside the feed pipe to the lower (dirt) cone. This immediate removal of the heaviest dirt particles from the coal cone reduces the danger of collision between coal and dirt particles which might cause the buoyant air bubbles to be detached from the coal particles. Water admitted under pressure in a tangential direction from the nozzle, *z*, imparts a rotary movement to the refuse in the lower cone, *a*, to enable residual coal particles to rise without contact with dirt particles. The rotary movement of the water in the lower cone, *a*, is not imparted to the main body of the flotation froth through the intervention of stationary baffles fixed radially between the rings, *x*, and the sides of the cone separator. The separated dirt settles in the pipe, *k*, on to the clack valve, *m*, which is operated by a lever, *o*, connected with a hinged lever, *r*, by the rope, *q*. The valve is actuated by the rotation of the cam, *t*, the amount of opening being governed by the turn-buckle, q^1. The opening of the valve releases the dirt into the seal, *l*, and also gives a pulsation inside the upper vessel, *ab*, and facilitates the free and continuous discharge of the coal through the throat, *c*.

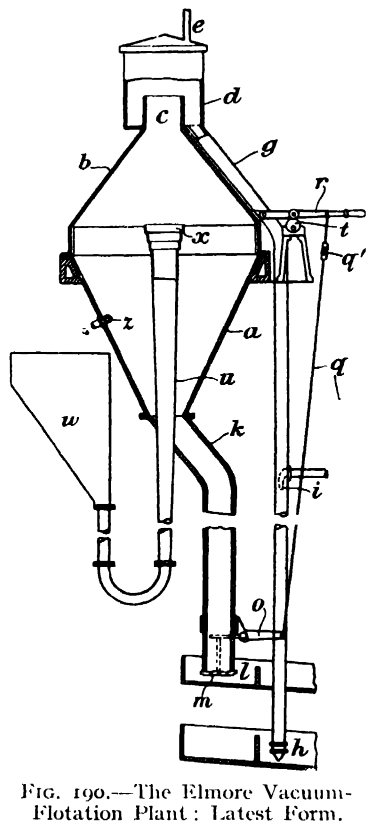

FIG. 190.—The Elmore Vacuum-Flotation Plant: Latest Form.

With some coals a voluminous froth tends to reduce the capacity of the appliance because of its slow discharge through the pipe, *g*. The velocity of discharge in such event is increased by admitting water under pressure through the jet, *i*.

The vacuum is applied from a vacuum pump through the pipe, *e*, at the top of the separating cone.

The new Elmore vacuum-flotation appliance therefore pays particular attention to removing all factors of design which tend to induce turbulent motion in the feed pulp. A turbulent motion tends to remove the air bubbles from large coal particles, and when turbulence is avoided, the size of coal floated may be considerably increased. The rotary currents and air admission in the lower cone

also give any settled coal another chance to float. It has been found under the new conditions that coal up to $\frac{3}{16}$ in. size may be floated. This is much larger material than can be floated by any other froth-flotation process and confers a great advantage to the one under consideration. The effect of the presence of larger coal particles in the floated coal is that the moisture content can be considerably reduced, thus again removing one of the particular drawbacks of froth-flotation practice.

It is an easy matter to apply a vacuum from the pump to the recovered coal and to reduce the moisture considerably by this treatment. The use of a comparatively small suction to wet coal effects a greater removal of water than the application of considerable pressures, and with the expenditure of considerably less power.

One of the advantages of the Elmore flotation process is that the power requirements are very low, since none is required for agitation and for the entrainment and distribution of air bubbles. Power for these purposes is required in most other froth-flotation processes. Only a small power expenditure is necessary to produce a vacuum and to mix the oil with the feed. To this important factor must be added the greater size of coal which may be treated, and the possibility of water reduction by the application of suction.

CHAPTER XXIII

MISCELLANEOUS COAL WASHING PROCESSES

MANY washers, the Baum and Rheolaveur, for example, are designed to handle large quantities of coal, and, with throughputs of over, say, 50 tons per hour their power consumptions fall within reasonable limits. Such washers are, however, not readily adaptable for small outputs, and in such circumstances simpler types of washers may be preferred. For example, at Cowpen Colliery, Durham, the slack used for boiler firing contained from 25 to 30 per cent. of ash and had a low calorific value, so that it was frequently difficult to maintain the necessary pressure of steam with such a fuel, particularly during the long operation of clinkering. A Hoyle washer of about 20 tons per hour capacity was erected to wash this coal, and the results proved most satisfactory. The washing of coal for beehive coke manufacture, or of a small quantity of a special brand of coal, or of the products of very small collieries, are further examples for which a cheap washer of low capacity is required. The capital and running costs should both be low in such cases. Some washers which are suitable under such circumstances have already been described, for example, the Robinson, the Blackett and the Elliott. Certain concentrating tables are also suitable for the purpose. Other washers of this class, as well as certain Continental and American washers, are described in this chapter.

THE GREAVES WASHER

The Greaves nut-washer is a movable-sieve jig, consisting of a steel tank of prismatic shape, in which a cradle forming the wash-box is supported from a lever with its fulcrum at one end of the outer steel tank. The wash-box has a screen plate on which the bed forms and is enclosed by steel plates forming the sides of the box. A cam-shaft fitted with two cams revolves in an oil box, supported on the outer steel tank. By the revolution of the cam-shaft the cams intermittently depress the free end of the lever to which the wash-box is supported, and the raw coal fed to the box through a shoot is subjected to pulsations, by the movement of the screen plate through the water in the outer tank. The violence of the pulsations is greatest at the feed end of the wash-box and decreases in intensity as the feed passes over the screen towards the discharge end of the box. The upper layers of clean coal in the bed flow over a weir, into the buckets of a scoop drainage wheel or of a small elevator, from which the drained coal is discharged into

a shoot, the drainage water flowing back into the outer tank. The dirt settles on to the inclined screen plate and moves towards a refuse-discharge gate. The refuse-discharge gate is formed by a section of the screen plate being hinged in the middle to a fixed support below it. The hinge is extended to form a lever, to the other end of which a balance weight is attached and projects above the level of the top of the wash-box. The balance weight is adjusted to keep the refuse gate closed when the bed is light, but when sufficient dirt accumulates on it, the gate is opened automatically and the refuse is discharged into the outer tank. From here the refuse is removed by a scraper elevator.

The washer fulfils the requirements of simplicity of design and, since there is no external water circulation, no power is absorbed by circulating water pumps. With a suitable design of the cams, the upward water current through the bed (during the downward movement of the screen plate) can be made to occupy a longer time than the downward water current and the harmful effects of suction are minimised.

One unit is of 15 to 20 tons per hour capacity. It is customary to divide the coal before washing into such fractions as 4 to 2 in. and 2 to $\frac{3}{4}$ in.

In the Greaves slack washer the design is the same as in the nut-coal washer, except that the washed coal is removed from the box by an unperforated scoop wheel and is discharged into a tank alongside the wash-box. In this tank the washed coal settles and is removed by means of a scraper elevator. An overflow pipe in this settling tank returns the clearer water from the upper levels to the outer tank of the washer.

The simplicity of design and the low power consumptions have recommended themselves to many users, and over 150 washers have been erected in Great Britain, with a total hourly capacity of 4,750 tons.

THE HOYLE WASHER

The Hoyle washer, illustrated in Fig. 191, is a jig of unusual construction. It consists of a tank divided into two compartments, of which one is the washing section and the other is a settling tank or washed-coal sump. The juxtaposition of these two sections makes the washer compact and renders the use of a circulating pump unnecessary. The washing compartment, A, has a sieve or grid plate, C, which is hinged at one side, K, and is connected to a lever system, M, and balanced by a weight, N. Below the hinged grid plate, a plunger, which is hinged at one side, J, is connected at the opposite side by a connecting rod, H, to an eccentric of variable stroke, F, actuated by a shaft, G. The plunger is given a reciprocating motion by rotation of the shaft, G. The plunger is provided with a series of valves, E, which open on the downstroke and close on the upstroke, thus giving a pulsating water current through the grid, C.

Coal is fed to the grid through a shoot, P, and is subjected to a jigging motion. The raw coal is classified, the lighter coal particles rise to the top of the bed, and the dirt settles to the grid plate, and gravitates towards its lower end. When the weight of dirt resting on the grid reaches a certain predetermined amount, the grid is depressed through the unbalancing of the weight, N, and the dirt is evacuated into the lower dirt compartment. Here it is collected by the screw, U, into the scraper elevator, V, for discharge.

The coal from the top of the bed is carried by the water stream over the adjustable weir, S, into the second compartment, B, where it settles. The washed coal is removed from the settling sump by the scraper elevator, T, which drags the coal over a drainage screen,

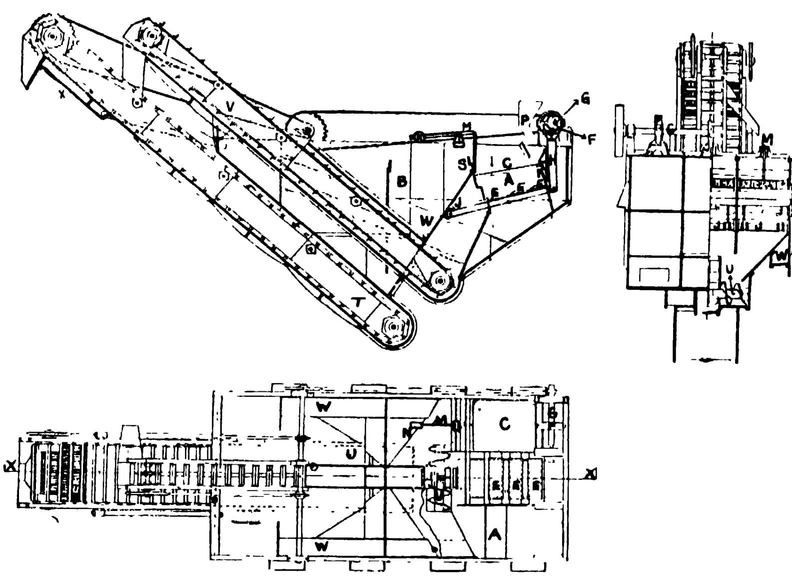

FIG. 191.—The Hoyle Washer.

X, set at a height which allows the drainage water to flow back to the settling tank by gravity. The drained coal is then loaded directly into wagons. Another view of the Hoyle washer, showing the scraper elevators, is shown in Fig. 192.

The simple water circulation is a particular feature of the washer. The water in the washing section, A, is forced by the plunger through the grid and overflows to the settling tank, where its velocity is reduced to allow the coal to settle. From this tank the water overflows into launders, W, Fig. 191, at each side of the tank, and returns to the underside of the grid. In this simple circulation of water, the Hoyle washer resembles the Sheppard nut-coal washer, which has a settling compartment adjacent to the washing section. The Hoyle washer, is, however, used for washing fine coal, and in the Sheppard fines washer a pump is used for water circulation.

The Hoyle washer is therefore a simple and ingenious modification of the ordinary jig washer. The dirt is automatically rejected, and there is no external water circulation. The working parts are few in number and are easily controlled. The power used is small, so that it fulfils most of the requirements already specified for a washer of low capacity. The standard washer has a capacity of 15 to 25 tons per hour. The one used at Cowpen Colliery, Durham, for washing coal for boiler-firing, deals with material less than $\frac{1}{4}$ in. in size. Float and sink tests on the products gave the figures recorded in Table 122.

TABLE 122.—RESULTS OF WASHING WITH A HOYLE WASHER

	Raw Coal.	Washed Coal	Refuse.
< 1·35 S.G.	50·6	90·0	6·4
> 1·35 S.G.	39·2	9·5	90·0
Loss	10·8	0·5	3·6

Allowing for the losses during testing (which in the case of the raw coal is an abnormally high figure) the sinkings in the coal were reduced from over 40 per cent. to under 10 per cent., which would probably give an ash content in the washed coal of under 10 per cent. The refuse contained about 7 per cent. of good coal. This is a larger figure than would be permitted to a modern large-capacity washer, representing as it does over 2 per cent. of the weight of the raw coal treated. Nevertheless, taking into consideration the simplicity and cheapness of the washer, the results are satisfactory for the purpose, for if the unwashed coal had been used for boiler firing, the combustible matter carried away with the large amount of clinker which would be formed would be several times greater than the loss sustained in washing.

Further results were given by Drakeley (*Trans.* Inst. Min Eng., 1919–20, 59, 71), and are recorded in Table 123 :—

TABLE 123.

RESULTS OF WASHING $\frac{3}{16}$ IN. TO 0 COAL IN HOYLE WASHER

	Raw Coal.	Washed Coal.	Refuse.
Ash per cent.	23·7	5·7	66·4
Floatings in 1·35 S.G. . .	65·6	91·3	7·6

Fig. 192.—The Hoyle Washer: Scraper Elevators

[To face page 142.

These figures also show a fairly high loss of coal in the refuse.

The Hoyle washer is also used for cinder washing—for example, at the Sheffield municipal household-refuse handling plant, where different grades of refuse are washed, and the recovered cinders are used for steam-raising or are briquetted and sold as a low-grade fuel.

THE NOTANOS WASHER

The Notanos washer employs a current of water down an inclined plane, and may be regarded as a concentrating table in that it has a partly riffled deck which is given an oscillating movement. Its length is, however, much greater than that of the usual type of concentrating table, so that it also resembles a trough washer in many respects. The washer is illustrated in Fig. 193. It consists of a long watertight steel trough built up of steel plates, angles and tees. The trough is slightly inclined to the horizontal, and is carried on chilled

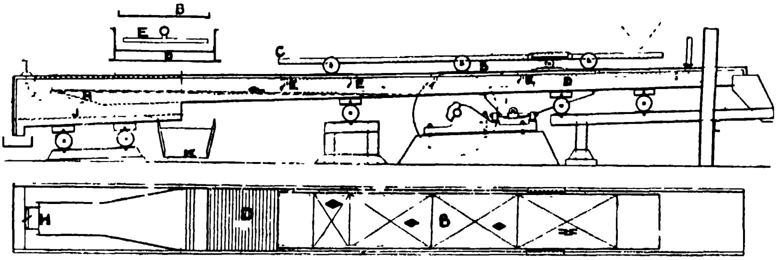

FIG. 193 The Notanos Washer. Elevation, Plan and Cross-section.

cast-iron rollers supported on channel-steel stools, the rollers being held in position by steel spindles coupled together by springs. The trough is actuated by means of "Marcus" gear. In the Marcus motion, the acceleration is almost uniform during the greater part of the forward stroke, so that the velocity reaches a maximum at a point near the end of the stroke. The velocity then falls suddenly until the motion is arrested. During the return stroke, the velocity rises rapidly to the same maximum as during the forward stroke, and is then gradually retarded until the motion is reversed. By reason of the momentum gained during the accelerating period of the forward stroke, the material near the surface of the trough continues to move forward during the greater part of the return stroke, so that movement in one direction is almost continuous. With the use of this mechanism for conveyors, a longer stroke (12 in.) and a slower speed of revolution (65 r.p.m.) are used than in most other types of oscillating conveyor.

In the washer illustrated in Fig. 193 the raw coal is fed on to the perforated tray, B, through which the undersize, through $\frac{5}{8}$ in., passes, and the larger coal passes over the end, at C, on to the riffled washing surface, D. Here the coal meets a supply of water from

the pipes, E, which is so regulated that the lighter coal particles are washed over the riffled surface and are carried over the adjustable weir, H, on to a draining sieve, which is oscillated by the same motion as the trough. The excess water passes to a launder and the washed coal to a shoot. The dirt falls to the floor of the riffled deck and is carried by the motion of the washer to the upper end of the trough, which is not riffled. The oscillation of the deck enables coal particles which may be entangled with the dirt, to be freed, and the current of water passing down the deck carries them towards the coal-discharge end.

The undersize coal falls to the upper part of the washing trough, where it encounters gentler currents of water from the pipes, E. The coal is carried to the lower end of the trough, H, and the dirt in the opposite direction. The power necessary when washing coal is said to be 0·8 h.p. per ton.

Although the washer illustrated can deal with both the nut and fine sizes of coal in the same trough, it is more usual to use a separate unit for each size. The different units in such a lay-out are placed side by side and are driven from the same shaft, but the length of the strokes are adjusted to suit the sizes of coal. A differential water supply for different sections of the trough is then unnecessary and each trough is supplied with a specified water current.

Details of the capacity and power consumptions of the Notanos washer are recorded in Table 124.

TABLE 124.—DETAILS OF NOTANOS WASHER

Width of Trough (ft.).	Power required (h.p.).	Capacity (tons/hour).	Water Circulated (gallons per minute).
4	7·5	10	200
5	9·0	15	290
6	12·5	20	380
7	15·5	25	470

THE MENZIES WASHER

The Menzies washer (or "hydro-separator"), illustrated in Fig. 194, was developed recently in America. It is essentially an upward-current classifier built of wood, with a simple water-circulating system. Raw coal is fed from the shoot, A, and enters the separating compartment, C, through the gate, B. An upward water current from the pump, T, passes through an inclined screen, H, into the separating compartment, C, and carries the lighter particles of coal upwards. The coal then passes through an opening, D, into a subsidiary compartment, E, and by a discharge valve, F, on to a

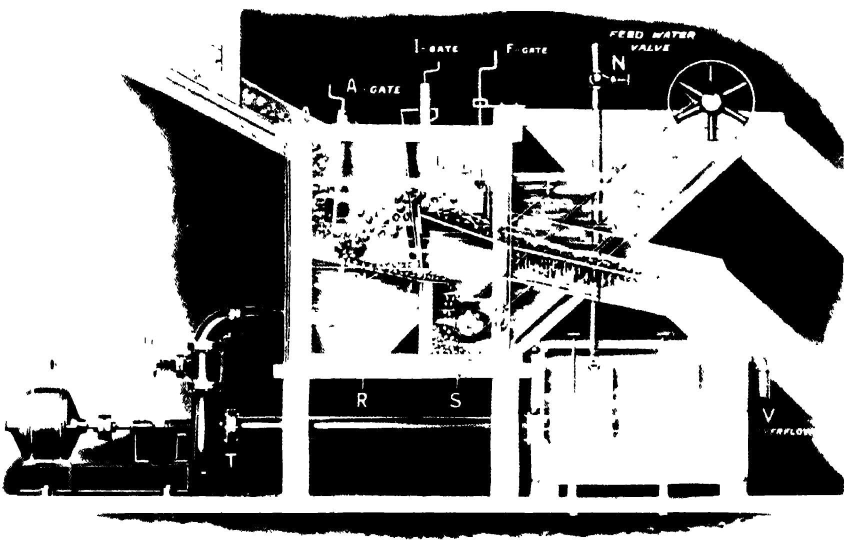

FIG. 104.—The Menzies Hydro-Separator.

[To face page 114.

drainage screen, which is totally enclosed. The coal passes over the screen into a shoot, but the water passes through the screen and is collected in the settling tank, M. Here the slurry settles in the spitzkasten, P and Q, and the clearer water is pumped off and returned to the separator. The strength of the current supplied is adjusted by means of the valve, O.

The dirt which remains on the inclined screen, H, works its way by gravity through the opening, J, and falls into the lower part of the subsidiary compartment. Here a scraper elevates it and discharges it into a shoot for disposal. The scraper works in a compartment full of water, but some drainage is possible above the water level. Make-up water is supplied through the pipe, N. The silt which collects in the bottom of the separator may be discharged through the valve, R, and two silt valves are also fitted to the spitzkasten, P and Q. When the refuse scraper jams, the lower part of the subsidiary compartment may be cleaned out through the valve, S.

The washer is of very simple design, and in this respect is reminiscent of the form in which the Bérard washer was introduced into England seventy-five years ago, or in which the Sheppard washer was built fifty years ago. It is known, however, that upward-current washers, if they are to effect an efficient removal of dirt from coal and of coal from refuse, should conform to certain simple requirements, namely, the production of uniform water currents over the whole area of the washer and the absence of eddy currents. Generally a slow and uniform helical current is produced in the washer which is made symmetrical about a longitudinal axis, a cone, for example. Moreover, it is usually arranged that the refuse is subjected to stronger water currents near the bottom of the washer to avoid undue loss of coal with the refuse. In the Menzies washer, however, the water is pumped in at one side, and the bed on the screen is subjected to the full force of the uneven and turbulent currents produced. The washing compartment is of irregular shape, and the raw coal is fed almost on to the screen. The Menzies separator, therefore, does not conform to many of the principles of upward-current classification which, as a result of forty years of experience with such appliances for coal washing, are considered to be desirable.

The washer is small, light and compact. Its capital and upkeep costs are probably low, and it is said to be easy to operate and to control. From considerations of its design a high working efficiency would not be expected from it, but, nevertheless, it may prove to be suitable for use for cleaning coal in a small works for boiler-firing purposes. A unit is said to have a capacity of 15 to 30 tons per hour, operating on any size of material from pea to barley (1 to $\frac{1}{8}$ in.), 3 to 5 h.p. being used by the pump and $\frac{1}{2}$ h.p. for the refuse conveyor.

THE HYDROTATOR WASHER

The Hydrotator process of coal cleaning has recently been developed in the American anthracite coalfields. Some experiments have also been carried out with bituminous coal. The first commercial plant has been erected at the Middle Creek Colliery, Pottsville, Pa. The washer is designed primarily to treat the slush or fines produced in washing the larger sizes of anthracite by other methods; the anthracite slush corresponds to the slurry produced from bituminous coal and is usually less than $\frac{3}{32}$ in. The Hydrotator unit is also suggested for use for general slurry washing and could be incorporated with existing types of washer which deal satisfactorily with larger sizes.

The Hydrotator unit may be used in three ways, as a thickener of the slush or slurry produced by wet washing methods; as an upward-current classifier for the slurry; or as a froth-flotation unit. A general purpose unit is illustrated in Fig. 195. It consists of a square wooden box with sides 4 ft. long, built on a concrete foundation. The slurry is fed to the unit through the feed funnel, *a*, below the quiescent water layer, *d*. The lower limit of the quiescent layer is the position of a pump-suction pipe with openings to remove water at a number of points across the full width of the cell. The water is drawn off by the pump, *e*, and returned to the cell through a delivery pipe which passes through the bottom of the compartment. The delivery pipe feeds four branch pipes, set at right angles to each other, the water being discharged through a number of jets in each branch pipe on to the bottom of the cell. An upward water current is therefore produced, acting fairly uniformly over the whole area of the cell from the bottom to the level of the upper suction pipe. The material which settles to the bottom of the cell is continually agitated by the jets. This method of producing an upward current is calculated to avoid the production of eddies.

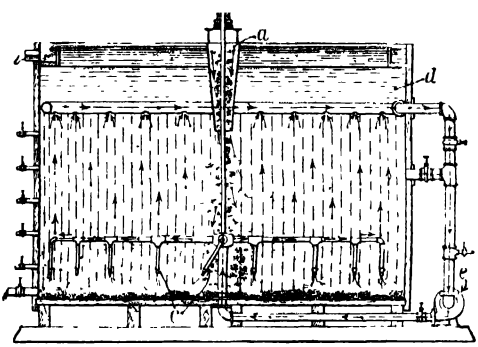

FIG. 195.—The Hydrotator Cell

When the cell is working as a thickener, its diameter may be increased to reduce the velocity of the upward current, and the floor of the cell may be inclined towards the centre to aid the concentration of the solids at that point. The finest particles rise to the level of the pump-suction pipe, and, after passing through the pump, are discharged on to the bottom of the tank, where they tend to be

trapped by the overlying, thickened solids. The relatively clear water overflows at the top for discharge. In anthracite preparation, a thickener is first used to concentrate the slush, which initially

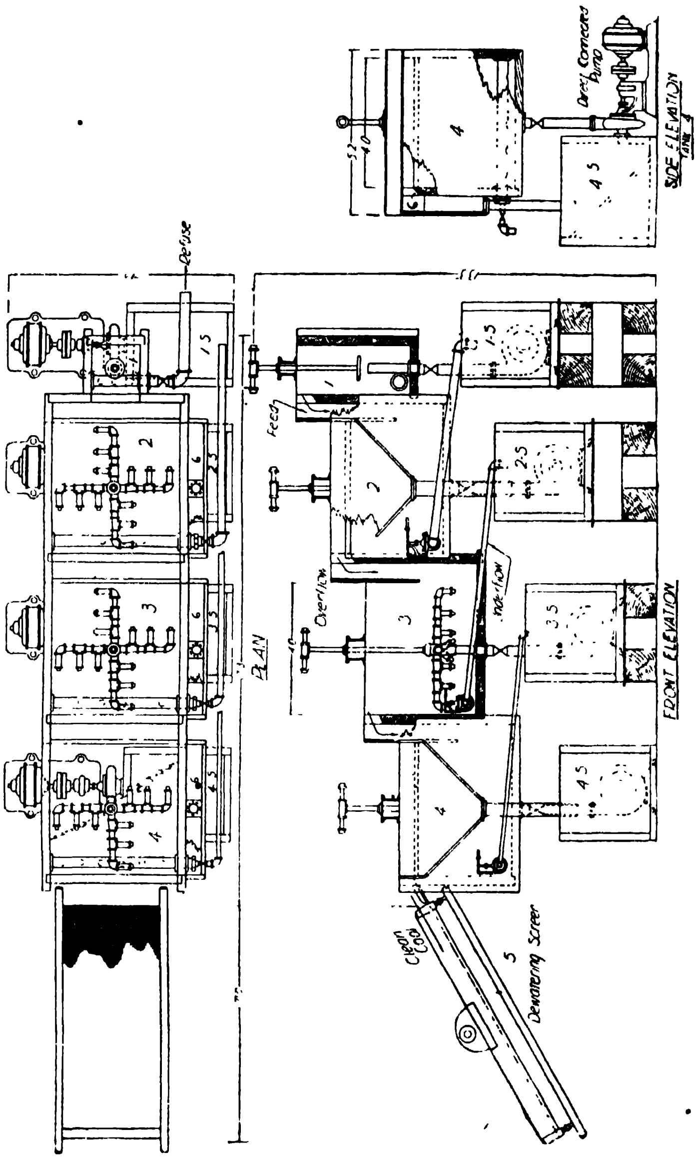

FIG. 196.—The Hydrotator Upward-current Classifier.

contains only 5 to 10 per cent. of solids. The thickener cell is usually worked to remove all particles finer than 100 mesh in the overflowing water, since the coal of this size cannot be recovered by the standard upward-current classifier.

As an upward-current classifier, a Hydrotator cell is fed with thickened slush (which is first screened over a 65-mesh screen if the finer sizes are very dirty) and the upward current is adjusted to carry the coal particles to an overflow at the top of the cell. Usually four cells are worked together, so that the overflow from one cell passes to the second one, and so forth (see Fig. 196). It is said that most of the recovery of coal takes place in the first cell, and the refuse (which may contain 20 per cent. of the recoverable coal) is discharged to the tank, 1-s, to be pumped to the refuse-cleaning cell. Here the coal is recovered by an upward current and the refuse is discharged from the bottom of the cell. In the second and third cells (3 and 4, Fig. 196) the middlings settle out and are circulated to the first and second cells (2 and 3) through their respective sumps and pump suctions. The final washed coal is passed over a de-watering screen of 65 mesh, and the particles smaller than this are removed with the water.

In this adaptation of the Hydrotator cell, the pump suction pipe in the upper section of the tank (Fig. 195) is not used, the water circulation being effected by allowing some of the water to overflow from each cell through a side opening, 6, Fig. 196, to the sump of the same cell. In cells 2 and 3, the pumps in the sumps 2-s and 3-s therefore deal with the overflow from their own cells and the underflow from the succeeding cells 3 and 4. The overflow is regulated to give the desired suspension of solids in the cells.

For such a plant of four cells, a space 24 ft. by 7 ft. by 13 ft. high is required and from 100 to 200 tons of material can be washed per eight-hour day. The power requirements are from 8½ to 13 h.p. for the circulating pumps and de-watering screen. Typical results for cleaning anthracite slush in this way are recorded in Table 125.

TABLE 125.—RESULTS OF CLEANING ANTHRACITE SLUSH (THROUGH $\frac{3}{32}$ IN.) IN THE HYDROTATOR CLASSIFIER

Size. Tyler Mesh.	Before Washing. Per cent by weight.	Before Washing. Ash per cent.	Size.	After Washing. Per cent. by weight.	After Washing. Ash per cent.
> 20	15·3	19·9	> 20	12·4	7·4
20–48	35·2	26·3	20–28	40·6	9·8
			28–48	20·8	12·7
48–65	9·1	33·3	48–65	25·2	20·3
65–100	7·4	35·4	< 65	1·0	28·7
< 100	33·0	44·8			
Average		33·7			13·0

The material smaller than 65 mesh is lost during de-watering, and in this test the amount so lost comprises 39·4 per cent. of the raw material. The material so lost has an ash content of about 44 per cent., and if the ash content of the rest of the refuse were high (over 70 per cent.), the average ash content of the total refuse would be much less than the refuse discharged from a Rheolaveur slurry washer, for example.

For the froth flotation of coal, a series of cells is used somewhat similar to that employed for upward-current classification. The coal is, however, fed to the last unit (4, Fig. 196), and the refuse is re-cleaned successively in the preceding units. The reagents are added in the pump circuit, to which air is also admitted. The water overflow is adjusted to give only sufficient upward current to carry over the floated coal, most of the water being discharged with the refuse. By controlling the amount of overflow, a combination of upward-current classification and of simple froth-flotation action may be used to vary the quality of the product. The usual flotation reagent is said to be crude tar oil in amounts of 0·1 per cent. of the weight of the clean coal. The refuse from the flotation unit may be crushed if it contains larger particles of coal than can be readily floated, and the fine coal produced may be recovered by froth flotation in the next unit (3). In this modification, the fine material through 65 mesh is not removed during dewatering and the percentage recovery of coal is therefore greater. The efficiency of cleaning is also much higher than in the upward-current classifier, as shown by the results in Table 126 for an anthracite slush, and in Table 127 for Alabama bituminous coal.

TABLE 126.—RESULTS OF CLEANING ANTHRACITE SLUSH IN HYDROTATOR FLOTATION UNIT

Size (Tyler Mesh).	Weight per cent.	Ash per cent. in Washed Product.
> 20	12·5	6·9
20–48	45·0	8·4
48–65	11·3	8·9
65–100	12·5	9·1
< 100	18·7	11·3
	Average	8·7

Comparing Tables 125 and 126, it will be noted that, whereas with upward-current classification only 1 per cent. of material less than 65 mesh was recovered, flotation of the same material yielded a recovery of 31·2 per cent. of coal through 65 mesh. Moreover,

TABLE 127.—RESULTS OF CLEANING BITUMINOUS COAL IN HYDROTATOR FLOTATION UNIT

Size (Tyler Mesh).	Weight per cent.	Ash per cent.		
		In Raw Coal.	In Floated Coal.	In Coal Cleaned by Jig Washer.
> 20	71·5	19·6	1·6	3·9
20–48	19·6	24·8	2·8	11·2
48–100	4·8	28·0	3·6	16·0
100–200	2·5	29·6	5·2	19·2
< 200	1·6	34·9	8·6	23·3
Average .		21·6	2·2	6·6

the mean ash content of the recovered anthracite was 8·7 per cent. compared with 13 per cent. by upward-current classification. The ash in the coarsest size of clean coal was only 6·9 per cent., although the fixed ash was "usually considered" to be 8 per cent. for this coal. On dewatering the product in a rotary vacuum filter (of the Oliver type) the water content of the floated anthracite was reduced to 19 per cent.

It will be seen, therefore, that the Hydrotator cell is of simple construction and may be used for a number of purposes. As an upward-current classifier it is not very efficient, but it is known (Chapter IV.) that coal should be carefully sized and that the water currents should be adjusted for the different sizes, to attain the best results in upward-current classifiers. Nevertheless, a considerable reduction in the ash content is made without sizing. Since all the material less than 65-mesh size is rejected, the yield of purified material may be low, and a large quantity of refuse material is produced of a type offering difficulties in disposal. The Hydrotator cell as an upward-current classifier would therefore be inferior to the Rheolaveur slurry washer in efficiency and more limited in use in a highly industrialised country like Great Britain (where effluents are not easily disposable), but it may prove useful in special circumstances. As a froth-flotation unit it is more efficient than as an upward-current classifier

THE MALECOT WASHER

The Malecot washer, which was devised in Belgium, resembles the Rheolaveur washer in some respects in feeding coal to a steeply-inclined trough, to the end of which an appliance is fitted, where the actual separation of dirt from coal takes place. An upward water

current in this member allows the dirt to settle but prevents the discharge of lighter coal particles with the dirt.

The separating appliance is illustrated in Fig. 197. A dam virtually forms the end of the inclined feed shoot and forces all the raw coal to pass into a vertical channel to which an upward water current is supplied. The heavy dirt settles against the water current and is discharged with part of the water supply through the lower inclined pipe. The lighter particles of coal and middlings are carried upwards by the water current and overflow into a collecting trough. The separating appliance therefore bears some resemblance to the rheo-box of the Rheolaveur washer. In the Rheolaveur washer, stratification of coal, middlings and dirt is produced partly by

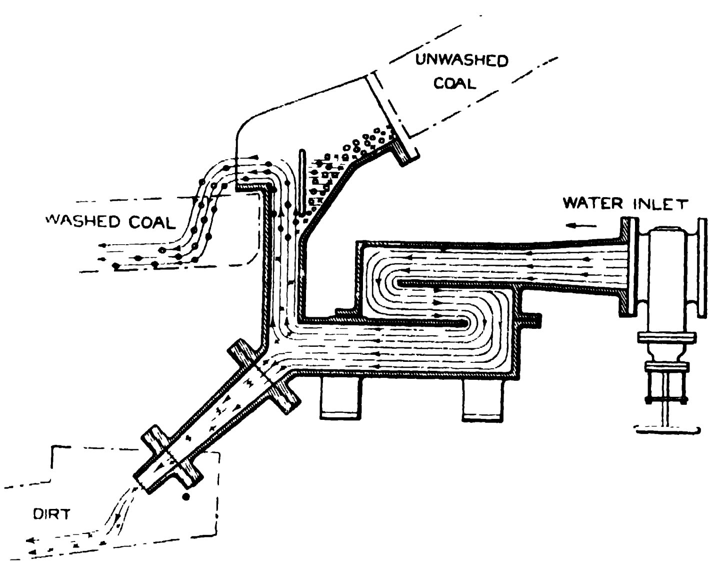

FIG. 197.—The Malecot Upward Current Washer.

alluviation in the steeply-inclined feed shoot, and partly as a result of the gradual reduction of the inclination of succeeding portions of trough, the employment of barrages, and modification of the cross-section in different lengths of the trough. Once the stratification is produced it is carefully preserved, and one of the functions of the rheo-boxes (apart from the removal of the refuse) is to prevent disturbance of the upper layers of coal and middlings. In the Malecot washer, however, after a rough stratification is produced in the inclined feed shoot, the coal is suddenly diverted downwards and meets a sudden upward water current. Obviously, the principles of separation employed in the Rheolaveur washer are not utilised to the same extent in the Malecot washer, and many of the refinements incorporated in the Rheolaveur, as a result of practical experience, are absent. It would appear to differ from the Rheolaveur washer

in that more reliance is placed upon the classification of coal and dirt by the upward water currents. Indeed, although troughs are used in the washer, and some stratification undoubtedly takes place in them, the main separation is performed by the upward currents and the washer may be regarded more as an upward-current classifier than as a trough washer.

In practice, Malecot grades the raw coal before washing, using a size ratio of 3 to 2. In washing nut coal, the upward current in the first classifying appliance is regulated to recover pure coal, which is collected in a transverse launder. The dirt and middlings fall against the upward current into an elevator which discharges the material into a second steeply-inclined trough terminated by a second upward-current appliance. From this unit the middlings are recovered in an overflow launder and the dirt is discharged into an elevator.

In the fine-coal washer, the design more nearly resembles that of a

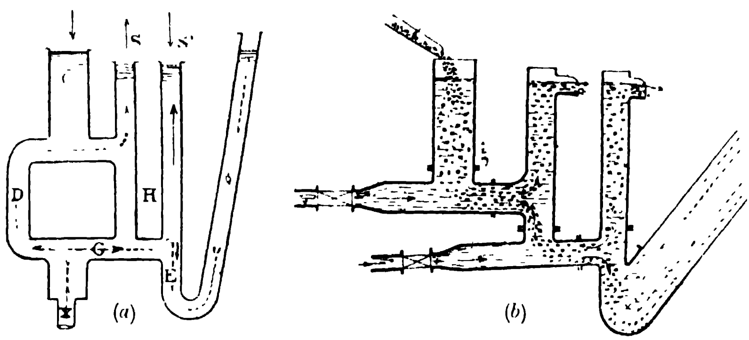

FIG. 198.—The Delcuvellerie Upward Current Washer.

Rheolaveur washer in that the upward-current appliance is fitted in an almost horizontal portion of the trough. The upper layer of the material is carried over the gap by the force of the water current. The dirt, together with some lighter particles, falls through the gap and meets an upward water current. The dirt continues to fall and the lighter particles are carried back again to the trough. For fine coals, three upward-current appliances are used. The first one rejects final dirt, and the second middlings and further dirt, which are re-treated by a third unit discharging dirt only. The pure coal product is recovered at the second unit and the middlings fraction at the third. A number of Malecot washers have been built in the central and southern coalfields of France.

THE DELCUVELLERIE WASHER

The Delcuvellerie washer was devised in France several years ago, but until recently it was not possible to use it on a commercial scale. It is an upward-current classifier but has not the simplicity

which usually attaches to this type of washer, due, partly, to an attempt to recover a separate middlings fraction. As illustrated in Fig. 198 (*a*), it consists of three vertical columns, and an inclined column for refuse removal. The raw coal is fed into the column, C, and the cleaned coal is removed through the column, S, and middlings through the column, S^1. Upward currents are produced in the various columns by supplying water from X. On meeting the raw coal at the bottom of the feed column, the water current from D is sufficient to carry the lightest particles up the column, S, and the residue is subjected to rewashing in the lower column, H. The material which finally settles in this branch meets a stronger water current at the bottom, and this stronger current causes the middlings

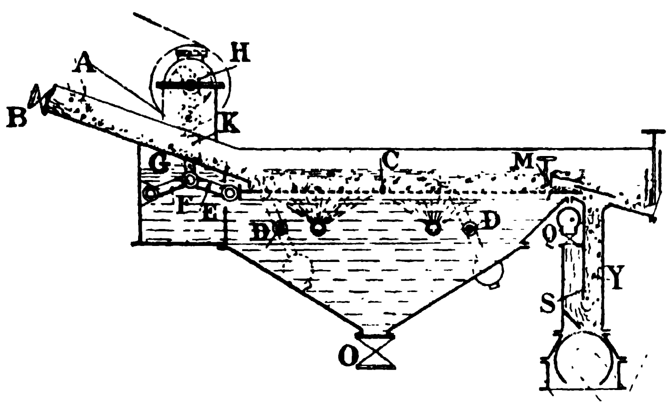

FIG. 199.—The Lequeux Washer.

to ascend the column, S^1, whilst allowing the dirt to settle through E to the dirt elevator.

In Fig. 198 (*b*) an improved arrangement is shown which allows better control by reason of the separate supply of water for the coal and middlings columns. A further arrangement (not illustrated) embodies the method of discharge used in the Rheolaveur washer. The raw coal at the bottom of the supply column meets a stream of water in a horizontal conduit in which a gap allows the discharge of the dirt, but the lighter layers are carried forward to a second gap surmounted by a column. The middlings fall through the gap and the coal is carried to the top of the column and is discharged.

THE LEQUEUX WASHER

Another French washer, the Lequeux, built at Gouy Servin, is probably unique in embodying the principles of trough washers, jigs and upward-current classifiers in one unit. It is illustrated in

Fig. 199. Raw coal is fed into an inclined shoot, G, at A, and meets a current of water from B. Some stratification of the material occurs before the feed is discharged on to the movable screen of a jig. The cam, H, periodically depresses the connecting rod, K, and the lever, E, transmits a stroke to the jig screen, C. The motion is balanced by the attachment of a second lever, F, to the connecting rod and to one side of the jig box. The depression of the connecting rod causes the screen, C, to rise slowly, producing, in effect, a slow downward current through the bed. The sudden release of the cam and the operation of the balance weights, D, draw the screen, C, sharply downwards, this motion producing, in effect, a sharp

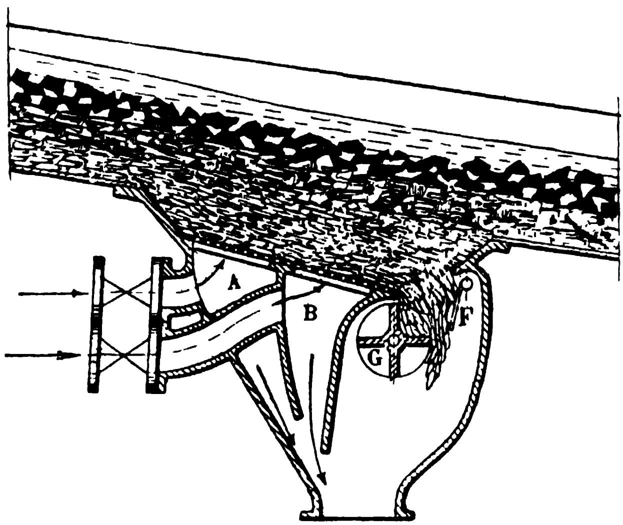

FIG. 200 — The Ranwez Multilaveur Rheo-box.

upward current through the bed. The material works its way across the screen, and the upper layers of material overflow at a weir and gate valve, M. The bottom layers of material pass under the gate to an upward-current classifier, Y. Water, admitted at Q, passes down one side of the partition, S, and rises on the other side to carry any light coal particles back to the overflow from the jig. The dirt which settles is discharged into an elevator. Not content with these various devices for ensuring classification and separation, the inventor directs upward water currents through the screen to keep the bed open. He also uses three similar units in series.

The washer does not recommend itself to those familiar with efficient jigs, trough washers, or upward-current classifiers. Nevertheless, by supplying the raw coal to a flow jig in a stream of water, in a suitable trough, the partial classification of coal and dirt before entering the jig might enable the throughput to be increased or the efficiency to be slightly improved.

THE MULTILAVEUR WASHER

The Multilaveur washer was introduced by G. Ranwez, and embodies many of the principles of separation used in the Rheolaveur washer. A steeply-inclined feed shoot is used, followed by lengths of troughs less steeply inclined. For nut-coal washers sealed rheo-boxes connected to elevators, and supplied with upward water currents, are used; for fine coal, rheo-boxes with adjustable discharge apertures are employed.

The washer differs from the Rheolaveur washer in the type of rheo-box used. The Multilaveur box is illustrated in Fig. 200. Instead of a steady upward current in the rheo-boxes, a pulsating or jigging current is produced. This type of current is claimed to make the bed more open and allow particles mechanically-entangled in the wrong layer to recover their true positions in a classification according to density differences. The pulsating current is applied to the two sections, A and B, alternately through a mechanically-operated valve placed at the junction of the two pipes supplying water to the compartments A and B. The valve-box and the pipes connected to sections A and B are in communication with an overhead water-tank, so that the pulsations are really intermittent upward water currents, which are transmitted to the material in the trough through suitable sieves. Part of the water passes downwards from A and B and into the trough through the refuse-discharge compartment. In nut-coal rheo-boxes the discharge valves, F, are controlled by hand, but are aided by the rotation of the discharger, G.

The addition of the use of intermittent upward water currents to the methods used to classify raw coal in a Rheolaveur type of washer seems unnecessary, and, indeed, more likely to disturb the classification than to improve it.

THE TRENT PROCESS

The Trent process for coal cleaning has been developed since the War and has found a number of uses in special circumstances. Essentially, the process makes use of the selective wetting of coal by oil and of mineral matter (dirt) by water. Finely-ground coal, on agitation with water and oil, is wetted by the oil and forms aggregations, but the dirt remains suspended and dispersed in the water. On passing the product over a suitable screen, the finely-divided mineral matter passes through the openings, but the aggregated masses of coal remain on the screen. Trent called the aggregated masses of coal "amalgam." About 30 per cent. of oil (or over 60 gallons per ton) is used, and the coal is ground through 200 mesh. It will at once be apparent that such a process could not be of general application in a country which is dependent on foreign sources for its oil supplies, unless the oil could be recovered in a more valuable form than prior to use.

In the experimental plant, the coals were ground in a specially-designed pulverising mill which consisted of seven horizontal steel cylinders bound together with iron hoops. Into each cylinder a number of narrow iron pipes were placed running from end to end, about half the cross-section of the steel cylinders being so filled. The whole apparatus was then rotated and fed with coal and water through bent feeding-pipes fixed in the feed end of each cylinder; the feed-pipes passed through a tank containing coal and water once in each revolution, and the ground coal passed out with the water at the other end.

The ground coal was then fed with water to the "amalgamator," to which oil was added, and the mixture was stirred. The "amalgamated" product was removed in the form of balls approximately $\frac{1}{8}$ to $\frac{1}{4}$ in. in size. The dirt removed from the coal remained suspended in the water. The "amalgam" of coal and oil contained mechanically-entangled water amounting to about 10 per cent., an amount which could be considerably reduced by kneading the material, or by passing it through a sausage machine. The efficiency of cleaning may be judged from the figures recorded in Table 128, which are results obtained on a laboratory scale. The coals were ground for six hours in a ball mill before treatment, so that the figures represent conditions which might not obtain in practice.

TABLE 128.—RESULTS OF CLEANING COAL BY THE TRENT PROCESS.

Coal.	Oil used. Gal./ton.	Ash per cent.		
		Raw Coal.	Cleaned Coal.	Refuse.
Anthracite culm	65	27·7	7·0	87·0
Pittsburg bituminous	80	12·5	6·0	88·0
Illinois bituminous	80	16·6	7·4	69·7
Indiana bituminous	80	9·9	6·3	86·2
Washington bituminous	80	22·6	13·6	85·0
California lignite *	80	35·1	25·7	75·9
Texas lignite *	80	33·5	18·1	94·2

* Carbonised at 500° C.

The efficiency of cleaning is only poor considering that the coal is ground to a very fine state, thus freeing the coal and shale particles from mixed particles. Equally good results could be obtained with fine coal in modern types of slurry washer without such grinding. In experimental work carried out at Sheffield University (see *Fuel*, 1927, 6, 146) it was found that, by separating the products on a screen, mineral matter is entangled in the amalgam, and much

better purification is obtained when the coal, after agitation with oil until it is not too strongly flocculated, is floated by aeration. The mineral matter removed in the tests recorded in Table 128 is, however, of very high-ash content. The oil used was a heavy fuel oil.

In the Sheffield experiments it was found possible to clean coals down to 1 or 2 per cent. of ash, and " fixed " ash was also removed. In ordinary washing practice it is not often practicable or desirable to crush coal before washing, but in these laboratory experiments the raw coal was ground in a colloid mill until the particles were less than $\frac{1}{1000}$ in. size. In these circumstances, the ash which for most practical coal-cleaning processes may be regarded as being " fixed " was freed, and removed from the coal by the application of the Trent process.

The particular interest of the Trent process is that the cleaned product can be readily dewatered until it contains only a few per cent. of water. It can be moulded into briquettes, and it is said that it can be pumped like a liquid. If an oil is used which distils below the decomposition temperature of the coal, the oil may be removed by distillation, and recovered, leaving a relatively clean powdered fuel. Alternatively, only a portion of the oil may be removed by distillation, leaving the pitch portion of the oil in the solid residue, which may then be readily briquetted.

Trent amalgam has been made in a large-scale plant at Alexandria, Virginia, U.S.A., and proved a popular fuel in closed stoves and kitchen ranges in the form of briquettes. It burns with a bright luminous flame, and gives a hot coke the shape of the original briquette. It has also been used in the horizontal retorts of the gasworks at Alexandria, for steam-raising purposes and for water-gas generators. When used in an open fire, the resulting smell of oil detracts from its otherwise pleasing performance. It is proposed to market the amalgam in 25-lb. " bricks," 14 by 8 by 4 in., wrapped in oil and moisture-proof paper. At Lapugnoy, France, a Trent process plant treats 20 tons of coal per hour, the ash content being reduced from 18 to 7 per cent. Briquettes are then made and used for locomotive firing. It has not found an extended market, however, on account of the cost of its preparation.

Distillation of the amalgam at different temperatures yields " cracked " oil and a quantity of gas containing a large proportion of ethylene. It has been suggested that this gas would prove a suitable starting-point for the manufacture of ethyl alcohol. The yields of ethylene at different retort temperatures are recorded in Table 129, with their equivalent alcohol yields based on the assumption that the whole of the ethylene could be converted into recoverable alcohol.

From these brief considerations it will be appreciated that the Trent process may be considered generally as a method for preparing clean coal briquettes, of high calorific value, for domestic

TABLE 129.—YIELDS OF ETHYLENE (AND ALCOHOL EQUIVALENT) IN GASES FROM DISTILLATION OF TRENT AMALGAM

Retort Temperature deg. C.	Percentage Ethylene in Gas	Total Gas Yield. Cubic ft./ton.	Equiv. Alcohol. Gal./ton.
400	23·4	750	3·1
500	17·0	4,400	13·8
600	8·8	10,850	17·3
800	8·9	19,800	32·0

and certain industrial uses. The efficiency of dirt removal, according to published figures, is not good when the fineness of division of the coal is taken into consideration. The removal of water from the cleaned product is an important result, but this is attained at the expense of using a considerable quantity of crude oils or tars, over 60 gallons per ton, which makes the process somewhat expensive to use.

Certain coals (Indian, for example, and some American) cannot be cleaned effectively by the usual methods of coal-cleaning, since the cleanest particles of coal have " fixed " ash contents of 10 to 20 per cent. or more. If it were practicable to grind such coals very finely (finer than 200 mesh) and cheap supplies of oil were available, a modified Trent process would be more effective than any other process, and there would be no difficulty in dewatering the product. The Trent process, it has been claimed, has made Rhode Island coals usable. Rhode Island, U.S.A., was situated some distance from a good coal supply, and the native coal was supposed to contain too high a percentage of " fixed " ash for it to be used efficiently until the Trent process was adopted.

GENERAL CONSIDERATIONS

In considering washers of low capital and running costs for small collieries, or for purposes for which only a low capacity is necessary, a balance should be made between the smallness of the running costs and the loss of coal in the refuse. A washer with a capacity of 20 tons per hour operating on a ten-hour day and removing 20 per cent. of dirt, containing 7 per cent. of free coal, would reject 2·8 tons of coal per day or 840 tons in a 300-day year. A more efficient type of washer would leave only, say, 2 per cent. of coal in the refuse, and the difference in the annual coal recoveries of two such washers would be 600 tons, worth, say, £300. This sum must be compared with the saving in interest on capital and running costs,

and it may indeed show that the cheapness of a washer is more apparent than real, for it amounts to 1*s*. 5*d*. per ton of the throughput. A washer like the Blackett, which can be used equally well for both large and small capacities, with equal efficiencies for all capacities, can be used under almost all circumstances.

CHAPTER XXIV

PAN-ASH SEPARATORS

THE coke usually found in the ashes of a boiler or producer consists of small particles of partly-burnt material that have passed through the grate bars or that have been removed with the clinker. Its quality is slightly inferior to that of pieces of the original fuel of corresponding size, and it may be contaminated with adherent incombustible dust. Carbonaceous matter may often be enclosed in fused particles of clinker, and many of the unfused ash particles may contain a proportion of unburnt carbon. In washing ashes, the object is to recover the free coke particles, the combustible matter in the remaining particles being of little practical value. The separation of these particles from heavy clinker is a matter of no difficulty, for the porous coke particles enclose quantities of air and have a low apparent specific gravity. The particles which are most completely burned are also fairly easy to remove, for they consist almost entirely of heavy ash.

About 4,500,000 tons of coke are used annually at gasworks in the manufacture of the producer gas used to heat the settings. From this amount of coke at least 1,000,000 tons of " pan ash " is produced, probably containing about 40 per cent. of coke. Assuming an average amount of $33\frac{1}{3}$ per cent. of recoverable coke (worth 15*s*. per ton), over £250,000 may be realised annually by the recovery of the coke from ashes of gasworks producers. The ashes from boilers contain appreciable quantities of carbonaceous material, produced by the mechanical fracture of the coal in the links of the grate as it travels forward, the average percentage of recoverable carbon in the ashes of mechanically-stoked boilers being from 10 to 20 per cent. In hand-fired boilers the gross weight of carbon in the ashes may be as high as 50 per cent., of which a large proportion is in the form of recoverable coke. It would therefore appear that the recovery of combustible material from ashes may often be a profitable undertaking at all large and medium-sized, or even at small, gasworks, and in many boiler plants where the number of boilers used is high. The washers or separators used for this purpose are usually called pan-ash separators.

At a gasworks with a bed of eight retorts, about 1 ton of pan ash is produced per day. From such a material the following products have been separated in a Robinson washer.

Pan ashes treated . .	$48\frac{1}{2}$ tons.	
Coke and breeze recovered .	21·2 ,,	= 43·1 per cent.

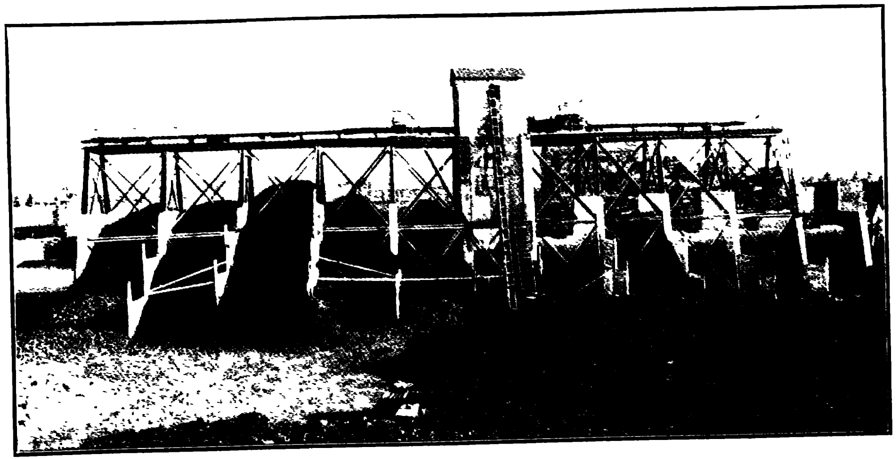

Fig. 201.—The Notatus Washer for Pan-Ash Separation.

Clinker and ashes . .	25·8 tons	= 52·4 per cent.
Silt	2·2 ,,	= 4·5 ,,

The pan ash from a bench of eight retorts would therefore yield about 150 tons of coke per annum, the coke being of fair quality. Since the washers adaptable for pan-ash separating are generally of low capital cost, have economical power consumptions, and are simple in design and control, it is frequently found that the cost of such a washer is soon recovered. The saving entailed is then very considerable, for the fuel is quite satisfactory for return to the producers. At one gasworks over 1,300 tons of coke and breeze were recovered during a period of six months. The recovered material contained about 28 per cent. of water when removed from the plant, and, after air drying for three days, the water content was reduced to 14 or 15 per cent.

A number of washers that have already been described for coal washing are also used for pan-ash separating. They are generally those of low capital cost which are simple in design and whose power consumptions are low. For example, the Hoyle washer is used at the Sheffield household-refuse handling plant. The raw material passes through cylindrical screens and is graded into sizes. The fine ash passes through ½ in. apertures in the first section of the screen and is loaded into a hopper for disposal without further treatment. The remainder of the household refuse is graded into two suitable sizes, and the oversize, which includes many tins, is hand-picked to recover these The two sized fractions are washed separately in Hoyle washers and the recovered cinders are used for steam-raising purposes on the plant, or are briquetted and sold as a low-grade fuel.

The Notanos washer has been erected at many gasworks in the larger towns of Great Britain, and also in Holland. For pan-ash separating a single trough is used for all sizes, since the necessity for highly efficient recovery of such material is not so great as that customary in coal-washing practice. In a washer used for pan ash at the Wandsworth Gasworks (*Gas World*, 1914 (Coking Section, December), 16) the capacity is 5 tons per hour. The washer is built on a structure 35 ft. high and 150 ft. long, which is divided into nine reinforced concrete bays, each of which is used to store a different size of product (Fig 201).

The pan ash from the producers is tipped on to a 2⅜ in. grid cover over the feed elevator, and the oversize material is broken up. The material is fed regularly to the buckets by a revolving star feed and is elevated to a ⅜ in screen which removes the undersize before the pan ash is delivered to the washer. Here the material is separated into coke and clinker. The coke is floated over a drainage screen and then over a series of grading screens, each size being dropped into its appropriate storage bay in the lower structure. The clinker delivered at the upper end of the washer is also sized

and each product is delivered into its appropriate bay. Four sizes of coke between $\frac{3}{8}$ and $1\frac{1}{2}$ in., and five sizes of clinker, are separated A reinforced concrete tunnel runs through all the bays to enable a horse and cart to pass beneath them. The material can be loaded from any of the bays through a door in the roof of the tunnel. This particular plant is driven by a 25 h.p. motor. The water is raised by a special gritty-water pump from a water clarification tank and returned to the washer.

As previously mentioned, the Robinson washer is also used for pan-ash separating, and is satisfactory for the purpose because of its simple design and ease of operation.

The Rheolaveur washer found one of its earliest uses in pan-ash, or cinder, washing when, in 1918, during an acute fuel shortage in France, thirty washers were erected at gasworks, chemical works, steelworks, collieries, etc., to recover coke from ashes. Subsequent to 1918, however, only five more Rheolaveur "cinder washers"

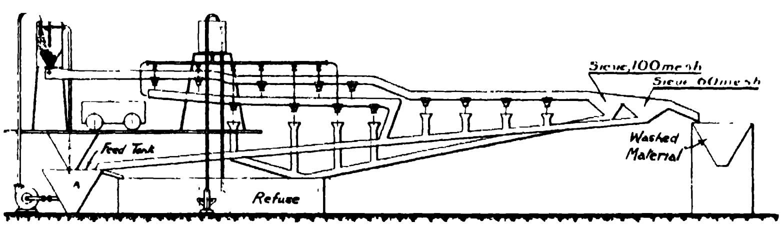

FIG 202.—The Rheolaveur Cinder Washer.

were erected. The washers had a small capacity, which varied from 5 to 20 tons per hour.

The Rheolaveur cinder washer is illustrated in Fig. 202. It consisted of two washing troughs with subsidiary collecting troughs to return rewash material to the feed tank, and the refuse to a settling tank. The raw material was fed to the conical tank, A, where it was well wetted with water before being pumped to the feed hopper at the head of the washing troughs. The lighter material—coke—was carried by the water stream from the end of the upper trough, and was drained by passing over a 100 mesh sieve, through which water containing the finest particles of dirt passed, and was collected in the dirt tank. The cleaned material was then passed over a 60 mesh sieve to remove the remainder of the water. Most of the finer particles of material passed through the sieve with the water, and were returned to the feed tank for re-circulation. The first five rheo-boxes in the upper trough rejected the heaviest dirt, and the remaining four rheo-boxes, intermediate material, so that only the lightest and cleanest coke passed from the upper trough. The heavy dirt rejected by the first five rheo-boxes was rewashed in the lower trough and any material recovered, together with the intermediate product from the remaining rheo-boxes of the upper trough, was returned to the feed cone for re-

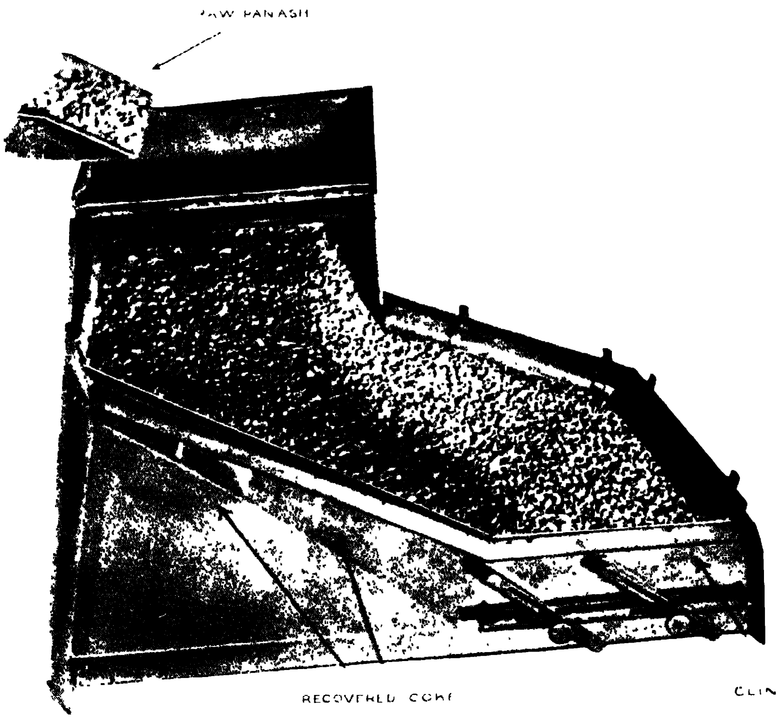

Fig. 203. The S J Separator. Pan-Ash Separation

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circulation. Water was supplied from an overhead tank to most of the rheo-boxes near the feed end of the washer, the upward currents produced preventing the passage of light coke particles through the rheo-boxes. The rewash material was returned to the washer for further treatment. The dirt settled quickly in the dirt sump, and the clearer water from the top was pumped to the overhead tank to feed the rheo-boxes.

The Birtley S.J. separator has been used experimentally to clean pan ash, and, if the sizing limits imposed are not too onerous,

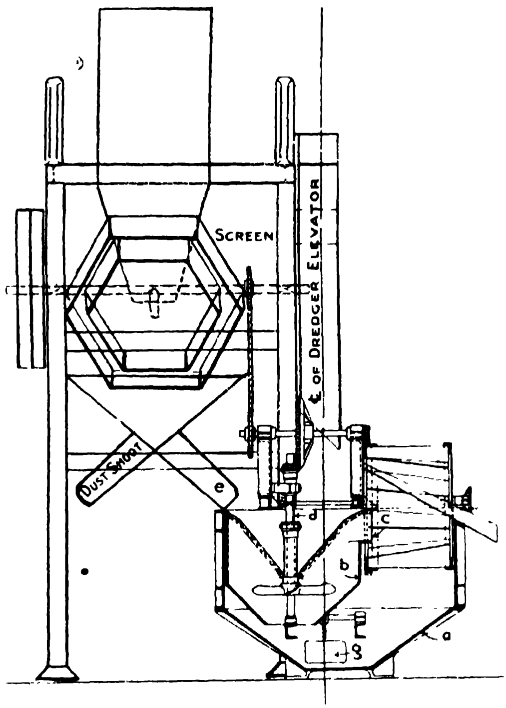

FIG. 204 – The Retriever Pan-Ash and Breeze Washer

an advantage is to be gained by cleaning pan ash in the dry state. The use of the S.J. separator for the separation of clinker from an oil gas plant is illustrated in Fig. 203. In this test, coke containing 18 per cent. of ash, and refuse containing 77 per cent. of ash, were recovered. By including some of the middlings with the clean coke, the ash content of the refuse was raised to 95 per cent., and the ash content of the coke was increased to 22 per cent.

The cleaning plants already described are those which have been designed primarily for coal cleaning, but which are easily adapted for pan-ash or breeze cleaning. Other plants, such as the Retriever, the Columbus, and the Ullrich, have been specially designed for the

separation of combustible material from pan ashes, coke breeze, and boiler or other ashes.

The Retriever Washer.—The Retriever washer, illustrated in Fig. 204, is used for pan-ash separating, for washing coke breeze, or for the recovery of combustible material from boiler ashes. The Retriever is an upward-current washer, in which the water current is produced by the circulation of water from an outer to an inner tank. The outer cast-iron tank, *a*, is rectangular with the lower portions of two of its sides inclined to form a trough. The upper portion of the inner tank, *b*, is circular, but the lower portion of it is of truncated-cone shape. In the inner tank a propeller is driven by a vertical shaft, *d*, through bevel gearing, and causes a circulation of water from the inner to the outer tank. In returning to the inner tank, the water produces an upward current, which is sufficient to float any coke not already raised by the eddy currents above the propeller.

In operation, the pan ash is first screened to remove the finest dust (which consists almost entirely of incombustible matter), and is fed to the washer by the shoot, *e*. The lighter coke is raised by the water currents, and overflows from the inner tank at the lip, *c*. The coke is freed from water in a revolving screen, *f*, and is discharged into suitable collectors The water passes through the screen into the compartment, *a*. The dirt settles through the lower opening of the inner tank, and is removed by the dredging conveyor, *g*. A view of a Retriever pan-ash separating plant is given in Fig. 205.

The Retriever washer may be built in the open on the ground level, or may be raised to a suitable height by steel supports to facilitate the loading of the products from hoppers. This washer has been erected at nearly thirty gasworks in Great Britain to wash pan ash or coke breeze. For pan ashes, a washer has a capacity of 2 to 3 tons per hour, and two men are required for labour. In three test runs on different plants, the percentages of coke recovered were 43·6, 45·1 and 39·1. At the Cleckheaton Gasworks (at which 223,000,000 cub. ft. of gas were made in the year 1924) 961 tons of pan ash were produced, from which 343 tons of coke (35·7 per cent.) were recovered. In addition to its value as fuel, carting and tip charges (at 2*s*. 6*d*. per ton) on this amount of coke were saved, and of the 370 tons of fine ash produced, 165 tons were sold at an average rate of over 1*s*. 6*d*. per ton. The washer was in operation on pan ash for only 320 hours in the year, the power consumption being about $1\frac{1}{3}$ h.p. per ton of pan ash treated. Make-up water at the rate of 10 gallons per ton of pan ash was required at a cost of 9*d*. per 1,000 gallons. The total costs, including labour, water, power, stores, and interest on capital, amounted to 1*s*. $6\frac{1}{2}$*d*. per ton of pan ash. These costs are high compared with the costs of coal washing, but this is because the capacity of the washer is only small. Never-

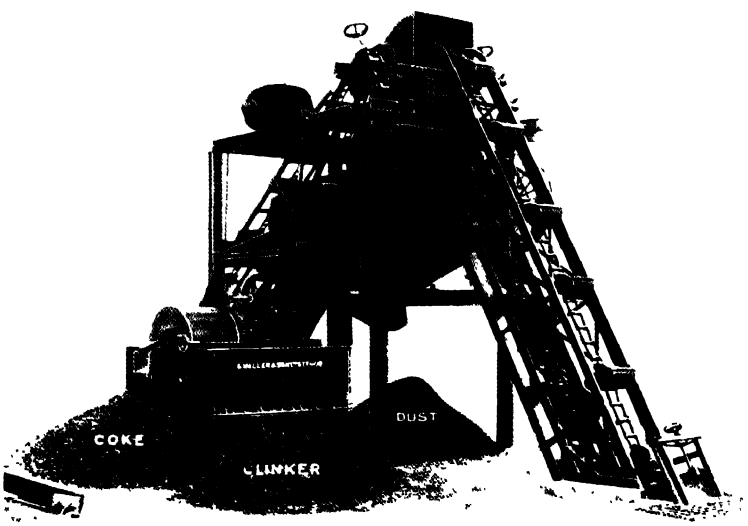

FIG. 205. View of Retriever Pan-Ash Separating Plant

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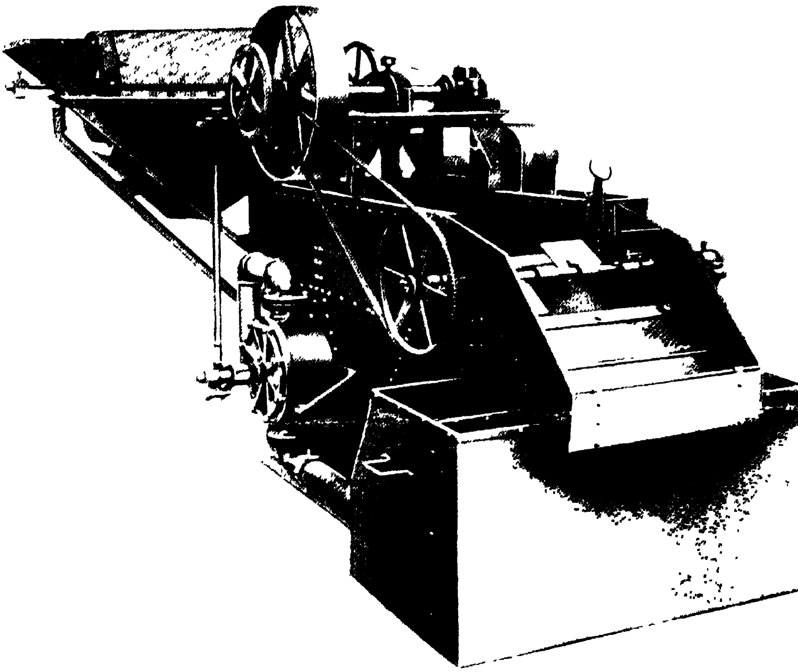

FIG. 207. A Jig Pan-Ash Separator

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theless, the year's working permitted a profit of nearly £500, which amply justified the practice of pan-ash washing.

In a test elsewhere on the ashes from a cupola core stove, 43·6 per cent. of coke was recovered.

The Columbus Pan-Ash Separator.—The Columbus pan-ash separator employs a medium heavier than water to effect a separation between clinker and coke. A suitable medium, of S.G. 1·21 to 1·26, is made by mixing water with clay or chalk (or other convenient substance) until the required effective specific gravity is attained. This suspension is contained in an inverted conical tank in which two worm conveyors are immersed. One conveyor reaches to the bottom of the tank to which the clinker settles, and the other is only slightly immersed below the upper surface of the separating medium, to remove the light coke which floats on the surface. In the separator illustrated in Fig. 206, an elevator feeds the pan-ash to

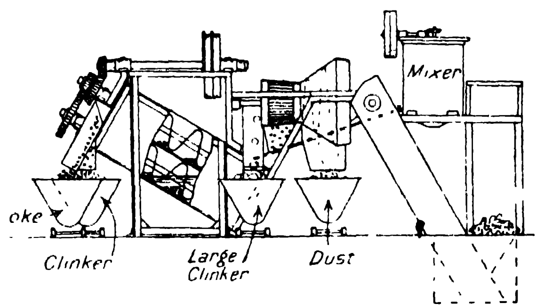

FIG. 206.—The Columbus Pan-Ash Separator.

a rotary screen, in the front section of which the dust is removed. In the second section, the medium-sized pieces of clinker are removed and are passed to the separator. The oversize passes over the end of the screen and, since it usually contains only a small amount of combustible matter, is not washed. The medium-sized material is separated by the medium of high specific gravity into coke, which floats on the surface of the medium, and clinker, which sinks. During the elevation of the separated materials from the tank, the drainage water runs back to the separator.

This simple separator, which requires no expensive machinery, no pumps, and no housing, has been erected at a number of small gasworks. The capacity of different units varies from 0·75 ton to 4 tons per hour; the smaller size requires 1 to 1½ h.p., and the largest size 3 to 4 h.p.

The Jig Pan-Ash Separator.—A jig pan-ash separator, marketed by Messrs. Silica Brick and Machinery, Ltd., is illustrated in Fig. 207. The pan-ash is fed to a cylindrical drum to remove the

ash less than 1 mm. size, and the oversize is fed to a wash-box. Here pulsations are created which cause the lighter coke to float until it is carried by the water currents over an inclined drainage screen. Drainage is aided by the use of a rotating shaft to which a series of plates are fixed ; these plates lift the coke from the overflowing water and deliver it again on to the screen. The clinker settles on the fixed screen of the jig box and passes under slides at both sides of the wash-box to the bottom. The clinker is removed from the wash-box by means of a scoop wheel fitted with perforated buckets and, after draining, is deposited into a discharge shoot. The water passes through the coke drainage screen into a tank, from which it is returned by means of a centrifugal pump to the wash-box. The pump is geared to an eccentric on the main shaft.

The washer is self-contained and needs no special foundations.

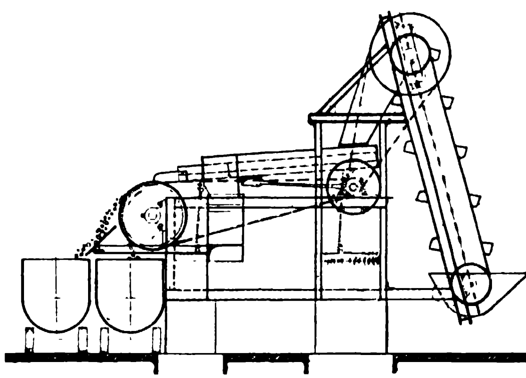

FIG. 209—The Ullrich Separator : Diagram of One Drum Plant.

From 1 to 4 h.p. are required, according to the capacity of the washer, which varies from ½ ton per hour upwards.

The Ullrich Dry Magnetic Separator.—The Ullrich process, unlike those previously described, does not depend for its action on differences in density between coke and clinker, but is based on the fact that ashes are attracted by strong magnetic fields which have no influence on coal or coke. This magnetic phenomenon is explained by the fact that the iron compounds almost invariably present in coal ash are reduced to free iron or converted into the magnetic oxide, or some other iron compound susceptible to magnetic attraction.

The Ullrich process was developed in the Krupp-Gruson works at Magdeburg, Germany. The British rights are held by Messrs. Chamber Ovens Ltd., London. One advantage of this process is that it operates on the dry material, and coke is recovered in the dry state. Moreover, the fine dust particles, which are usually removed before treatment in wet separators, can be treated by

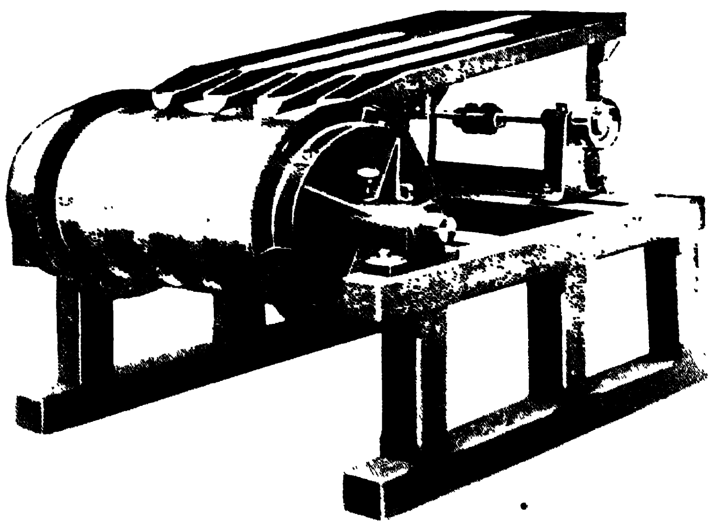

FIG. 208. The Ullrich Dry Magnetic Separator

[To face page 467.

the magnetic process as well as the larger material. The lay-out of the plant, with its absence of pumps, settling tanks and drainage screens, is simple and compact, and requires only a small floor space. The fine dry ash separated is claimed to be especially suited to the manufacture of slag bricks.

An Ullrich magnetic separator is illustrated in Figs. 208 and 209, which give a view of a single magnetic drum and an elevation through a one-drum plant capable of dealing with ½ to 2 tons of ashes per hour. The drum is provided with one to four magnetic fields, produced electrically. The ashes are fed uniformly to the drum by a series of shaking trays. The clinker and ashes are attracted by the powerful electro-magnets in the drum, and adhere to the drum surface for about half a revolution. As the drum revolves the coke particles are early projected from the surface of the drum, whereas the ash is retained in contact for a longer period, and only leaves the surface when the gravitational forces exceed the force due to the magnetic attraction, and cause the material to fall. The coke and the ashes are thus thrown off the drum surface separately, and are collected in tubs or other suitable collecting devices. In most cases it is desirable to screen the ashes into two or more grades before subjecting them to the magnetic treatment, and to use a magnetic field of different intensity for each grade. To facilitate the feeding process a picking belt may be added to the equipment to allow the largest material to be hand-picked. The whole unit may be mounted on wheels, and may be moved to different parts of a works, if desired. The power requirements per ton of ashes treated are stated to be from 1 to 1⅓ h.p. for the production of the magnetic field, and ¼ to ¾ h.p. for the drive

At a power station at Stettin, Germany, the ash from the end of the boiler grates falls down shoots on to a conveyor which discharges the material into a hopper. From the hopper, the clinker is crushed to pass a 1½-in. screen and is elevated, weighed automatically, and delivered to a second hopper which feeds the magnetic separating plant. The feed from this hopper passes over shaking screens, from which the oversize (which is usually free from coke) is rejected The screened product then passes through four outlets on to a drum with four elements producing the magnetic field. The coke recovered may then be loaded on to the coal conveyor for return to the boiler house. About 8 tons of coke are recovered daily. In five months, during which careful records were kept, 4,872 tons of ashes were treated with a total power consumption of 6·7 h.p. per ton. From these ashes 1,218 tons of coke were recovered (25 per cent.) with a calorific value equal to 80 per cent. of the calorific value of the coal used. The profit for each ton of coke recovered was estimated to be about 12*s*.

From one sample of pan ash, 57 per cent. of coke, with a calorific value of 10,600 B.Th.U. per lb., was recovered, and from another the yield was 53·4 per cent. Tests carried out at Krupp-Gruson works

gave the following results : From boiler ashes, 34·3 per cent. of coke recovered, of which 21·6 per cent. (of the original ashes) was less than $\frac{1}{2}$ in. size ; from core-drying stove ashes, 52·7 per cent. of coke, with 23·3 per cent. less than $\frac{1}{2}$ in. ; from producer ashes, 66·3 per cent. of coke, with 34·6 per cent. less than $\frac{1}{8}$ in. These figures show very well the advantage which the dry magnetic process has over wet processes for treating boiler ashes or material in which fine particles of coke are found. Over 150 plants have been erected at power stations, gasworks, etc.

CHAPTER XXV

THE DEWATERING OF COAL: DRAINAGE HOPPERS AND DEWATERING SCREENS

THE dewatering of coal is an important branch of coal-washing practice, more particularly when applied to the preparation of coal for coke manufacture. The water retained by wet coal is present as a film on the surface of each coal particle, and may also be retained by capillary forces in the interstices between the particles. With large particles of coal the ratio of the total surface area to the weight is small, and the percentage quantity of water retained as films on the surface of such particles is also relatively small. When the size of the particles is small, as in slack which all passes through a $\frac{5}{8}$ in. screen, the ratio of the total surface area to the weight is much greater than for nut coal, and the quantity of water retained as films on the particle surfaces is correspondingly increased.

The spaces between nut particles are so large as to preclude the retention of considerable quantities of water by capillary forces, and on standing in drainage hoppers for a short period of time, dewatering of even small nuts may be readily accomplished. On loading into a hopper, the water weakly retained in the spaces between the coal particles rapidly drains away. The total time necessary for drainage is governed mainly by the time required for the water, draining from the upper layers of coal to lower levels, to be removed from the lower levels before the coal is loaded into wagons.

With nuts of comparatively regular shape, unit volumes of specified sizes (viz., 2 to $1\frac{1}{2}$ in., $1\frac{1}{2}$ to 1 in., or 1 to $\frac{1}{2}$ in.) have the same weight, or, in other words, the percentage of free space is the same. When a hopper contains material which is not closely-sized and the ratio of the largest to the smallest sizes is great (as in coal of $\frac{5}{8}$ in. to 0 size), the volume of free space is much reduced, since the smaller particles fill in the spaces between the larger particles. Moreover, the ratio of surface to weight of unit volume becomes very large compared with a similar ratio for a graded nut size, and the aggregate weight of the films of water round the particles attains appreciable proportions. The juxtaposition of numerous particles whose superficial area is great, also allows the retention of water in the interstices between the particles by capillary attraction.

Fine particles of clay material retain water to a greater extent than coal particles of the same size, and, if present in sufficient quantity, may act as an impermeable membrane and cause drainage to cease. If active drainage from the hopper is not immediately renewed, considerable quantities of water, drained from the upper

layers of coal, may accumulate above the impermeable " membrane." When drainage from the hopper has been suspended for some time, an attempt to renew the natural drainage by poking at the hopper base may achieve the object, and the water may then flow out in a powerful stream and flood the whole of the drainage floor. Such an occurrence provides a dramatic proof of the resistance to drainage which clay particles in small coal may offer.

Theoretical Considerations.—Before describing the methods adopted for the dewatering of washed fines, it is pertinent to consider the physical aspect of the retention of water by small coal. When a number of spherical particles are arranged so as to give the maximum percentage of free space between the particles (an arrangement known

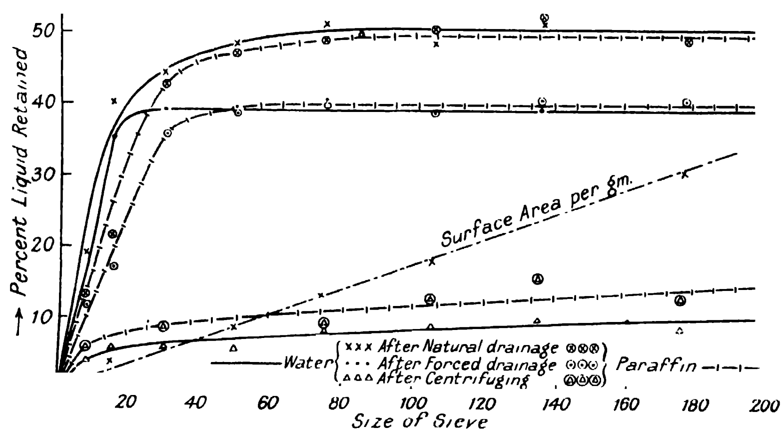

FIG. 210.—The Dewatering of Graded Sizes of Coal by Natural Drainage, Forced Drainage, and Centrifuging.

as loose piling), the percentage of free space is 47·6 per cent. of the total apparent volume occupied. When the spheres are agitated and caused to occupy the minimum apparent volume (an arrangement known as pressure piling), the percentage of free space is 26·0 per cent. Regular diamond-shaped particles give 50 per cent. of free space in " loose piling," but the free space is *nil* under conditions of " pressure piling." Coal particles are of irregular shape, and the amount of free space in a " loosely-piled " mass of particles (as in a hopper) cannot therefore be calculated.

A block of coal, 50 c.c. in volume, weighs about 65 gm., and if a volume of 100 c.c. were filled half with coal and half with water, the percentage weight of the water in the mixture would be 43·5 per cent. Actually, in experiments we have made, it was found to be 50 per cent. for closely-graded sizes of coal less than 50 mesh ($\frac{1}{100}$ in.) size, corresponding to a free space of 57·5 per cent. of the total apparent

volume. The results of these experiments are recorded in the upper curve of Fig. 210, which gives the amount of water retained by various sizes of small coal after natural drainage. The sizes chosen were 5-10, 10-20, 20-40, 40-60, 60-90, 90-120, 120-150, and 150-200 mesh I.M.M. The second curve shows the percentage of paraffin (by volume) retained by graded sizes of coal under similar conditions.*

The procedure adopted was to wet a weighed sample of coal thoroughly by agitation with water (or paraffin) in a small churn and to pour the wetted coal into a funnel lined with wetted filter paper, allowing the material to drain until continuous dripping ceased. The amount of liquid retained was then determined by direct weighing.

Curves I and II (counting from the top of the graph) show that graded coal of sizes from approximately $\frac{1}{100}$ in. to $\frac{1}{400}$ in. retain the same amount of liquid (water or paraffin) after natural drainage. The amount of water retained is therefore independent of size (or extent of surface) between these limits. For graded coal of size greater than 50 mesh ($\frac{1}{100}$ in.) the water drains away more readily than from the smaller sizes, and only 20 per cent. of water is retained by 5 to 10 mesh ($\frac{1}{10}$ to $\frac{1}{20}$ in.) coal after natural drainage.

The effect of forced drainage was then studied by "jigging" the funnels in filter-funnel stands. This action tended to induce "pressure piling" by reducing the free space between the particles to the minimum. The percentage of water and paraffin (by volume) retained by the different sizes of coal is shown in the second series of curves (III and IV) in Fig. 210. The reduction in the amounts of liquid retained is approximately the same for both paraffin and water. The two curves, III and IV, are of the same type as the curves for natural drainage, I and II, although the amounts of liquid retained are smaller for every size. There was a noticeable difference in the physical condition of the coal fractions of 40 to 60 mesh size and less, which yielded, after "jigging," a hard compact cone which resisted breaking up; the cones produced from the fractions over 40 mesh size readily broke up on touching. This physical condition of compacting was found for all the graded sizes which retained about 40 per cent. of liquid or, in other words, in which the spaces between the particles were completely filled with liquid. The apparent cementing action of the liquid gives an indication of the strength of the capillary forces causing its retention.

The condition of packing of the particles obtained in this series of "forced drainage" experiments approximates to the conditions obtaining on a jigging dewatering screen, on which the particles tend to pack by "pressure piling."

A further series of experiments was carried out to study the

* The amount of water retained is expressed as the percentage weight (or volume) of water in 100 gm. of the mixture, since this is the usual method adopted in practice. The amount of paraffin retained is expressed as the percentage volume of paraffin in 100 gm. of mixture.

retention of liquids by coal after centrifuging. A laboratory centrifuge was used, in which the funnels containing the wet coal after forced drainage, were centrifuged at a speed of 1,000 r.p.m. for 4 mins. The effective radius of the centrifuge was 7 cm. The centrifugal force, f, of a body moving in a circular path of radius r, is $\frac{v^2}{r}$. Compared with gravity, g,

$$\frac{f}{g} = \frac{v^2}{rg} = \frac{N^2 r}{25}$$

where N is the number of revolutions made per sec. (r, g and f being expressed in C.G.S. units). The centrifugal force was therefore

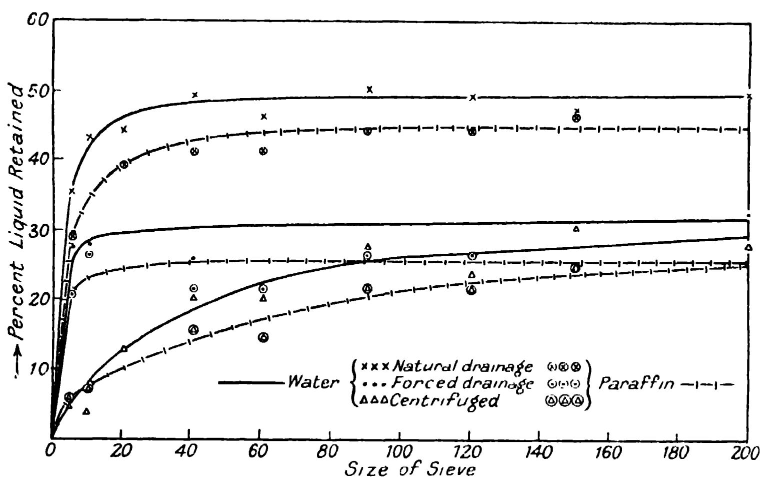

FIG. 211.—The Dewatering of Through Sizes of Coal by Natural Drainage, Forced Drainage, and Centrifuging

78 times the force of gravity.* The results of these experiments, using paraffin and water as the wetting liquids, are recorded in the third series of curves, V and VI, in Fig. 210. In these experiments there is the first indication given that the percentage of water retained by coal may be affected by the surface area of the coal particles as well as by the size and total volume of the interstices. The dotted line (curve VII) is introduced to indicate the relative surface areas of 1 gm. of the different grades of coal. (For the sake of argument the particles are assumed to be cubes, but the only purpose of the curve is to show the relative surface areas of different sizes.) It is clear that the amount of water retained after centrifuging is not directly proportional to the extent of surface area, although it increases with decrease in the size of coal. It is more probable that the slightly greater amount of water retained by the 120 to 150 mesh size, compared with the 60 to 90 mesh size, for example, is due

* Or about the same as in a 5 ft Carpenter centrifuge running at 300 r.p.m.

to the greater hindrance to water removal due to the more tortuous path followed. On the other hand, it is certain that centrifuging overcomes, more or less, the capillary forces retaining water in the spaces between the coal particles. Assuming that the water is retained entirely as a film on the particle surfaces, the film thickness for the 150 to 200 mesh size would be 0·00015 cm. It is interesting to note that more paraffin than water was retained by each size of coal, presumably due to the greater attraction exerted by the molecules at the coal surface for the non-oxygenated liquid.

Similar experiments were then carried out for the "through" sizes of coal, *i.e.*, coal which had been ground so as to pass through 5, 10, 20, 40, 60, 90, 120, 150 and 200 mesh. Thus the through 5 mesh coal contained particles of all sizes less than $\frac{1}{10}$ in., but none greater than $\frac{1}{10}$ in. The amounts of water and paraffin retained by these samples of the coal after natural drainage, forced drainage, and centrifuging are recorded in Fig. 211. For natural drainage the curve showing water retention is almost identical with that obtained for the graded sizes. About 50 per cent. of water was retained by the sizes with a smaller maximum size than 50 mesh, but more liquid was retained by the smaller "through" sizes. In such smaller sizes the filling up of the spaces between the larger particles by smaller ones would allow capillary forces to operate for a lower maximum size of particle. For paraffin the volume retained was about 45 per cent. (expressed on 100 gm. mixture) for the smaller sizes; this seems to indicate more effective filling up of the spaces between the larger coal particles by smaller ones in the presence of paraffin, due to its lower coefficient of friction or lower viscosity compared with water.

After forced drainage by "jigging," the amount of water retained by the smaller through sizes was about 32 per cent., as compared with 39 per cent. for the graded sizes. This again indicates the closer packing possible with through sizes. Only a few sizes of coal were examined after such treatment, but even for the coarsest sizes, through 5 and through 10 mesh, the amount of water retained was almost 30 per cent. The volume of paraffin retained under the same conditions was 25 per cent. for all sizes from through 5 mesh to through 120 mesh. Hence the curves illustrating the amounts of liquid retained, after reduction of the volume of the spaces between the particles through pressure packing, do not show points of inflexion at the 50 mesh size as they do for the graded sizes, and liquid was retained by capillary forces for all the sizes of coal tested. The minimum amount of water retained was 28 per cent. for through 5 mesh ($\frac{1}{10}$ in.) size, and this probably indicates the limit of dewatering coal of size less than 5 mesh, or smaller, by any process producing pressure packing of the particles, for example, dewatering jigging screens. In practice it is found that slurry (which is approximately through $\frac{1}{10}$ in. size) may be dewatered to about 30 per cent. by jigging screens.

The curves showing the amounts of liquid retained by the through sizes, after centrifuging, show points of inflexion at about the 40 mesh size, when 21 per cent. of water is retained. Sizes smaller than 40 mesh show a gradual increase in the amount of water retained the smaller is the size, until, for the through 200 mesh size, it is about 30 per cent., or practically the same amount as is retained after forced drainage. The amount of paraffin retained by the through 200 mesh size is also practically the same after centrifuging as after forced drainage. For the smallest size of coal tested, therefore, a force nearly 100 times as great as the force of gravity does not remove much more liquid than simple jigging, thus indicating the great strength of the forces retaining liquid at a coal surface. With sizes greater than 40 mesh, centrifuging effects an appreciable reduction in the amount of water retained, compared with forced drainage, the reduction amounting to 24 per cent. for both through 5 mesh and through 10 mesh size, although it is only 8 per cent. for the through 40 mesh size, and less for still smaller sizes.

It would therefore appear that, on a small scale, centrifuging affects a considerable dewatering for sizes greater than 10 mesh ($\frac{1}{20}$ in.) size, but is comparatively useless for sizes of 40 mesh and less. In practice the limits of usefulness of centrifuging may be considerably less than in these small-scale experiments, for the through 5 mesh sample, for example, was prepared by gently crushing larger coal so that the amount of very small coal would not be great. Through 5 mesh coal produced in practice would probably contain more very small particles, and the dewatering of such coal by centrifuging might therefore be inefficient. Scoular and Dunglinson, in fact (*Iron and Coal Tr. Rev.*, 1924, 104, 1112), refer to "the failure of the centrifugal method of drying coal after the flotation treatment"; their coal contained only about 13 per cent. of material coarser than approximately 15 mesh I.M.M.

The fact that centrifuging overcomes more or less the force of capillary attraction, and removes the liquid held by this force in the interstitial spaces between coal particles, was shown in the following way. When water is dropped on to a heap of fine coal it will not spread, but paraffin, on the other hand, spreads easily. When sufficient paraffin has been added to powdered coal its slightly brown colour becomes black and glistening, giving a visual impression of complete filming of the particle surfaces. A sample of fine coal was spread in amounts of 10 gm. on a glass plate, and various additions of paraffin were made in drops from a small dropping pipette. The results obtained for coarse coal (20 to 40 mesh) showed that 3 per cent. (by volume on weight of mixture) of paraffin was necessary to film the coal surfaces completely, and about 5 per cent. was necessary for coal through 20 mesh size. Such amounts give calculated film thicknesses of 0·0001 to 0·00025 cm. This probably represents the thinnest complete film of liquid paraffin

possible on coal.* A similar calculation of the film thickness of paraffin on coal after centrifuging shows a variation from 0·00035 for 60 to 90 mesh coal, to 0·00020 cm. for 150 to 200 mesh size. The corresponding figures for water are 0·0003 to 0·00015 cm. It would therefore appear that the liquid retained by coal after efficient centrifuging is held by strong surface molecular forces, and its removal can only be effected by the application of heat, or by the preferential wetting of coal surfaces by oils to displace the water to the interstitial spaces, from which it may be more easily removed by simple mechanical means.

Some further experiments made on the time necessary for dewatering by natural drainage are also of interest. These results for graded coal are recorded in Table 130.

TABLE 130.—THE NATURAL DRAINAGE OF GRADED SIZES OF COAL

Size. (Mesh I.M.M.)	Per cent. of Water in Wet Coal after Drainage for :—			
	5 min.	1½ hr.	3 hr.	20 hr.
5– 10	22	20·5	19·5	19·5
10– 20	38	36	35·5	35·5
20– 40	45	43·5	42·5	41
40– 60	46	44·5	43·5	42·5
60– 90	50	48·5	47·5	47·5
90–120	46	44	44	43
120–150	46	44·5	43	43
150–200	46	45	44·5	44·5
< 200	46	44	43	43

The coal contained 2·0 per cent. of ash; about 50 gm. of coal was contained in a funnel which was covered to obviate air drying. The results show that natural drainage was complete in about 5 min., and that long-continued drainage had only a negligible effect in further dewatering. Similar experiments for "through" sizes are recorded in Table 131.

These results are of a similar type to those previously recorded. They show that, in a thin layer, all sizes of coal rapidly drain to the limiting figure of natural drainage. In practice, a hopper may contain several hundred tons of coal, and a long drainage time is neces-

* Although it is possible to have a monomolecular film of gases on solids, or of liquids on liquids, the evidence that a monomolecular film of liquid may be produced on a solid is less substantial. It may be possible in froth-flotation processes where the filming liquid is agitated with the solid in the presence of another liquid (water). Where no agitation in the presence of a free liquid takes place, the film thickness is more usually equivalent to the diameters of several thousands of molecules.

TABLE 131.—NATURAL DRAINAGE OF THROUGH SIZES OF COAL

Size. (Mesh I.M.M.) Through	Coal A.		Coal B.	
	Per cent. of Water in Wet Coal after Drainage for :—			
	5 min.	18 hr.	5 min.	18 hr.
5	36	30·5	37	35
10	44	41	39	38
20	45	41·5	45	43·5
40	50	47	44	42·5
60	47	45	47	45
90	51	48	49	46
120	50	47	47	43
150	48	47	55	51
200	59	55·5	51	49·5

sary to allow the water to drain from the upper layers to the lower layers and to be removed.

Results obtained with coarser sizes of coal after natural drainage for 5 min. are of interest. They are as follows :—

Size (in.)	Per cent. Water.
$\frac{1}{2}-\frac{1}{4}$	4·8
$\frac{1}{4}-\frac{1}{8}$	8·2
$\frac{1}{8}-\frac{1}{16}$	10·7

These results indicate the small amounts of water which remain in the coarser sizes of coal after natural drainage. Dewatering by natural drainage is therefore quite satisfactory for sizes greater than $\frac{1}{10}$ or $\frac{1}{8}$ in., and additional means for dewatering are unnecessary.

It is well known that the presence of small particles of clay-like material considerably reduces the efficiency of dewatering by natural drainage. O. Schäfer (*Stahl und Eisen*, 1925, 45, 44) records results for the dewatering of raw slurry, and slurry cleaned by a froth flotation process, in a laboratory-scale hopper. The raw slurry would have an ash content of about 25 per cent., and the washed slurry would contain only about 8 per cent. of ash. The percentages of water retained after drainage are recorded in Table 132

These figures are of the same type as those previously given, but as a comparative series they illustrate the effect of the presence of clay particles in hindering dewatering.

Drainage Hoppers.—Drainage hoppers are commonly used to permit dewatering by natural drainage as well as for the storage of washed coal. They are usually built of concrete to hold from 20 to

TABLE 132.—PER CENT. OF WATER RETAINED BY RAW AND WASHED SLURRY AFTER NATURAL DRAINAGE

Size.		Per cent. Water Retained.	
Mm.	Approx. I.M.M. Mesh.	Raw Slurry.	Washed Slurry.
2–1	6–12	8·6	5·8
1–$\frac{1}{2}$	12–25	33·3	29·7
$\frac{1}{2}$–$\frac{1}{3}$	25–37	35·7	34·2
$\frac{1}{3}$–$\frac{1}{4}$	37–50	43·8	36·1
$\frac{1}{4}$–$\frac{1}{5}$	50–62	44·1	38·1
$\frac{1}{5}$–$\frac{1}{6}$	62–75	45·4	38·9
$\frac{1}{6}$–$\frac{1}{8}$	75–100	45·4	45·1
$<\frac{1}{8}$	$<$ 100	50·4	46·0

several hundred tons of coal. In Humboldt, Rheolaveur, and many other types of washery, the washed fine coal passes from the wash-boxes to a washed-coal sump from which the coal is removed by a drainage elevator. This consists of a series of perforated buckets joined by links in an endless chain, the elevator being inclined at a suitable angle to prevent the water drained from one bucket from dropping into the one below it. The drainage elevator discharges on to a scraper conveyor or some other device to feed the coal to a series of drainage hoppers.

A number of difficulties are experienced in practice with drainage elevators feeding from a sump. The finer particles of coal tend to choke the perforations in the buckets and to prevent drainage. If a washer has been standing idle for even a short interval, slurry settles in the washed coal sump and, on restarting, the buckets become partly filled with slurry which blinds the perforations. It therefore frequently happens that the material discharged on to the scraper conveyor or, in some cases, directly into the drainage hopper, contains excessive quantities of water. If the capacity of the drainage hoppers is not great, and the drainage time therefore small, dewatering may be a very inefficient process.

An important factor in the practice of dewatering nut coal is to avoid the production, by breakage, of smalls which would fill up the spaces between coal particles, and hinder drainage. In the plants built towards the end of the last century, Baum loaded nut sizes of coal into hoppers already filled with water, so that the water overflowed as the hopper became filled. The slow settling of the coal particles through the water in the hopper prevented breakage. When one hopper was filled with coal, the stream of washed nuts was deflected into another, similar, hopper, whilst the coal in the

loaded hopper was allowed to drain before unloading. In later plants Baum passed the washed nuts over a fixed drainage screen and loaded the coal into empty hoppers by special loading shoots which reached almost to the bottom of the hopper.

A number of attempts have been made to improve the effi-

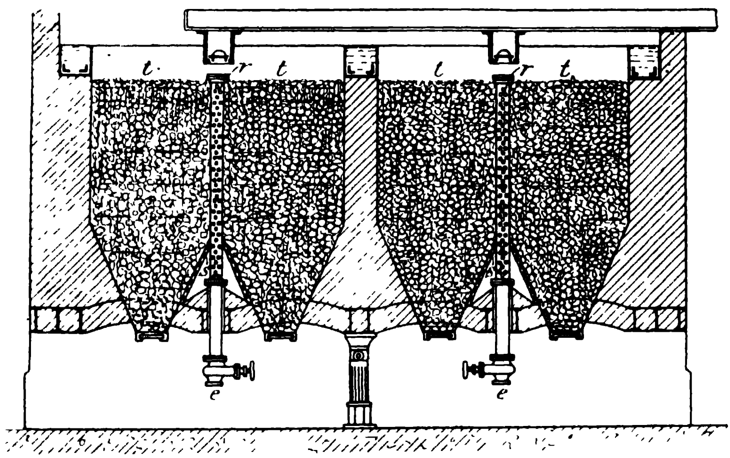

FIG. 2 2.—Drainage Hoppers (1891).

ciency of dewatering in the drainage hoppers. In 1891, Baum used a hopper divided into three or four sections and arranged two or three perforated pipes vertically so that water could be removed from all levels of the hopper. This is illustrated in Fig. 212. The

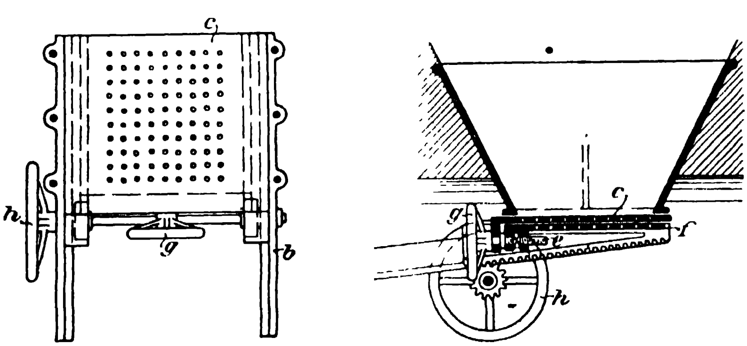

FIG. 213.—Perforated Plate Bottom of Drainage Hoppers.

inner sides of the lower tapered ends were also perforated to allow further outlets for the drainage water. The perforated pipe, *r*, was fitted with a waste valve, *e*. These hoppers were erected at a large number of Baum washeries in Germany, and later a number of Schüchtermann and Kremer, and Humboldt washeries were fitted with similar devices. These hoppers suffered from the general dis-

FIG. 214 View of Base of Drainage Hoppers with Rotating Base Plates

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advantage that the perforations of the drainage pipe were liable to choke, and, therefore, to stop the drainage.

The design of the base of hoppers is an important feature. Many hoppers have a rectangular perforated plate, or double plates as in Fig. 213, at the bottom. In Fig. 213, the hand wheel, *g*, is used to bring the perforations of the two perforated plates in line; the handwheel, *h*, is fitted with a pinion which operates a rack to which the perforated plates are fixed, so that they may be moved to one side to unload the hopper. When the perforations in this type of base plate become blocked they are difficult to open. The extent to which drainage is impeded may be shown by rapidly moving the pinion handwheel. Drainage is accentuated by this movement.

A better type of drainage base is that illustrated in Fig. 214. The base consists of a strong circular plate set horizontally; above it is a metal cylinder which is adjustable by a hand lever to give any desired distance between its lower end and the base plate. By

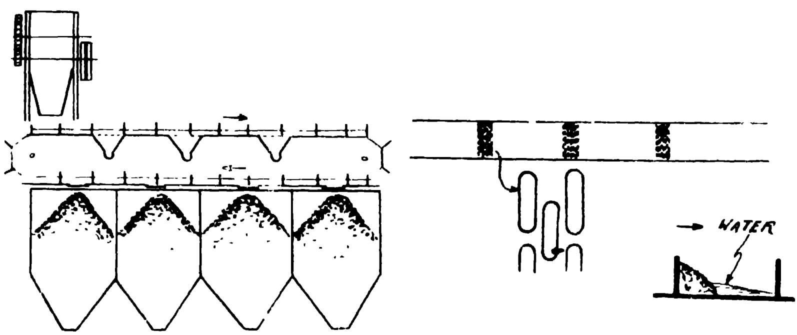

Fig. 215.—Rheolaveur Dewatering Device.

raising this cylinder, a quantity of coal runs out and rests on the base plate; the spaces between the particles of this coal take the place of the perforations in the type previously described. The drainage water overflows from the circular base plate into a launder. If drainage should cease or be restrained, the loose cylinder may be moved upwards and downwards rapidly. This is usually sufficient to permit drainage to be continued, but, if this fails, a rod may be inserted without loss of coal and new drainage passages opened up. To discharge the hopper after drainage, the base plate is rotated, and a plough fixed to guide the coal into a shoot.

It can safely be said that dewatering by natural drainage is an unsatisfactory procedure when a great excess of water is carried into the hopper, as frequently happens when coal is fed directly to the hopper by a drainage elevator from a washed coal sump. A number of devices have been introduced to prevent the passage of large quantities of water into the drainage hoppers.

One such device is the use of knocker bars, which are hinged to the elevator casing and are long enough, when in a horizontal

position, to touch the outside edge of the buckets. As the buckets move upwards, the knocker bars are raised until no longer supported by the rim of a bucket, when they fall on to the bucket below. The impact is sufficient to clear some of the perforated holes and so accentuate drainage. This simple device has proved effective in a number of washeries.

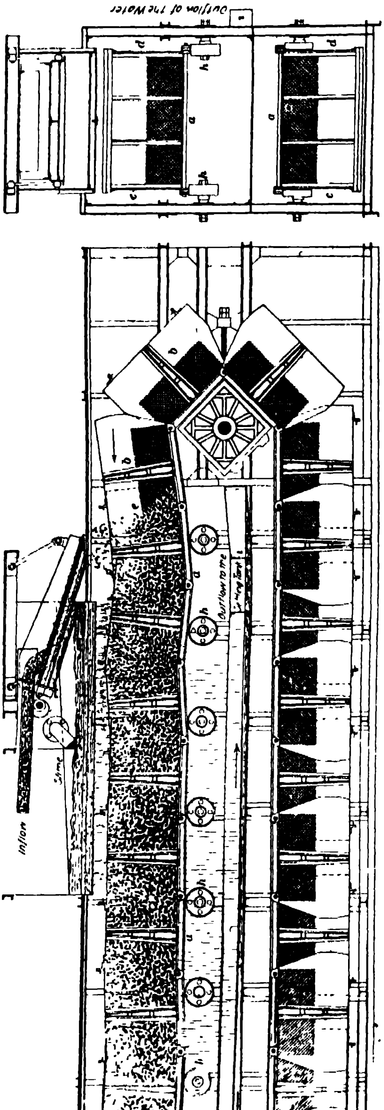

FIG. 216.—Diagram of Baum Drainage Conveyor.

Another device, which has been used in Rheolaveur washeries, treats the coal when it has been discharged on to the scraper conveyor feeding the drainage hoppers. It was found that a quantity of water was forced out of the wet coal as it was moved along by the scraper bars; the water collected in front of the charge of coal moved by each bar. This liberation of water from the wet coal is probably due to the pressure packing of the coal particles due to mechanical disturbance induced by the pressure and motion of the scraper. To remove this water, the base plate of the conveyor is perforated at intervals with elongated holes, the maximum dimension of which is equal to the minimum size of the coal. As the charge in front of each push plate of the scraper passes over the perforations, the freed water, with only a small amount of coal, passes through. Wedge-wire screens are sometimes used instead of perforated plates. By this means considerable quantities of excess water are removed from the coal. The arrangement is illustrated in Fig. 215 (Lecocq, *Fuel*, 1923, 2, 268), showing the drainage elevator, the scraper conveyor with three dewatering sections, and four drainage hoppers underneath it; the arrangement of the drainage openings, and a diagrammatic view of the extrusion of water in front of one push plate, are also shown.

FIG. 217. View of Baum Drainage Conveyor

[To face page 481.

The maximum size of a drainage hopper fed directly by a drainage elevator from a washed coal sump, is limited by the practical difficulties of dewatering. A large number of drainage hoppers are a very expensive item of washery construction, and this has led to attempts to dewater coal before loading into hoppers, which may then be built of large capacity for storage purposes only.

Baum Drainage Conveyor.—After Baum had tried numerous variations in the construction of drainage hoppers, in 1903 he introduced a drainage band for the dual purpose of dewatering and conveying the coal. The drainage band or conveyor, illustrated in Figs. 216 and 217, consisted of perforated plates, *a*, hinged one to the other, and carrying at the middle of each plate a double vertical partition, *b*. The partitions were constructed of perforated plates strengthened with angle irons, and separated slightly from each other to allow water to run between them. The sides, *c* and *d*, of the conveyor were also built of perforated plates. The conveyor was, in effect, a series of boxes hinged one to the other at the middle of the base plates. The links were made very strong to enable the conveyor to carry a load of two tons of coal per yard of its length. The movement of the conveyor was obtained by the use of rectangular drumheads driven through gear wheels from a belt-driven pulley. The conveyor passed over supporting rollers, *h*, spaced at such a distance apart that the conveyor sagged under the load of coal between one roller and another. The effect of this sagging was to press the coal between the vertical partitions, *b*, when the conveyor passed between the rollers, and to open the partitions when it passed over a roller. Through this action the coal was alternately compressed and released, and drainage was facilitated of the intermittent squeezing to which the coal was subjected. In the diagram (Fig. 216), a portion of the conveyor in its horizontal position of travel is shown. After the conveyor had carried the coal horizontally it conveyed the coal up an incline (Fig. 217) to a suitable height for loading into hoppers.

The method of charging the conveyor allowed the coarsest coal to be dropped on to the base, with the smaller material above it, and the slurry on the top. This layering was obtained by the use of swinging sieves placed above the conveyor. The washed fine coal was carried in a stream of water on to the swinging sieves, which allowed the water and finer sizes of coal to pass through, the coarser coal passing forward over the sieves on to the conveyor at *e* (Fig. 216). The finer coal, which had passed through the upper swinging sieve but remained on a lower one, was dropped on to the coarser coal. The finest coal and the slurry from the slurry sump were added on the top of these two layers and spread evenly over them. As may be observed in Fig. 216, the water which drained from the coal on the drainage band was collected in a trough underneath the filled section of the conveyor and carried to the settling tank, *i*, from which it

was passed to settling tanks to be clarified for re-use. The conveyor moved slowly, at a speed of about 8 in. per minute, to allow sufficient time for drainage during the travel of the coal to the storage bunkers.

This drainage conveyor was installed in Baum plants built in

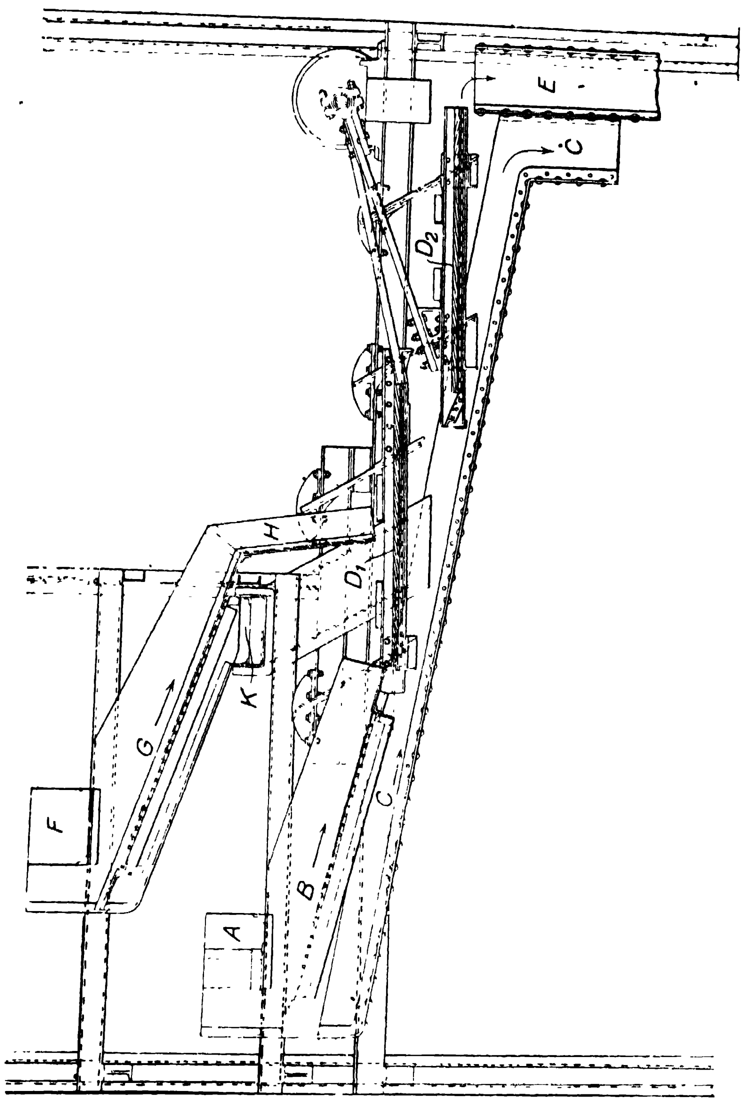

FIG. 21[illegible].—Dewatering Screens (Simon-Carves).

Great Britain by Simon-Carves from 1903 to 1914. Although a fairly efficient dewatering was effected, there were a number of disadvantages associated with its use which eventually led to its replacement by a simpler appliance. Although of very robust construc-

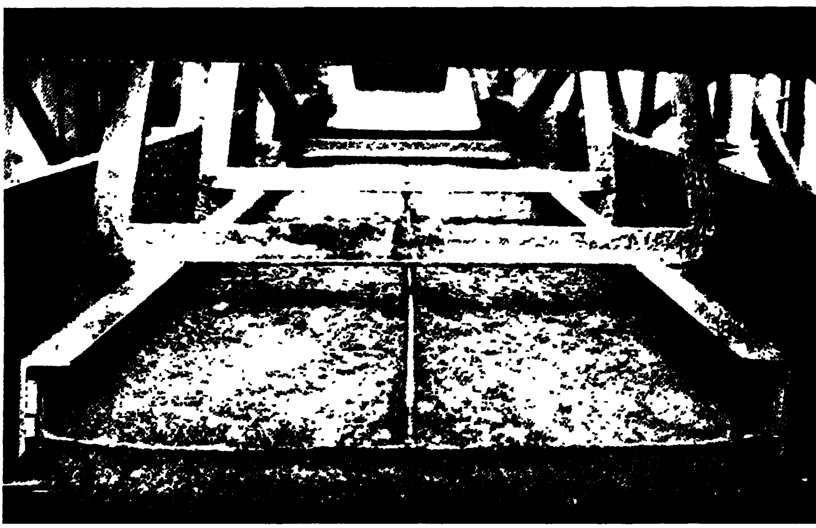

FIG. 219. View of Slurry Dewatering Screen (Simon-Carvers).

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tion, mechanical breakdowns were inevitable and repairs presented many difficulties. Moreover, its efficiency was not always high when dealing with certain types of coal. For example, it was frequently found that, in spite of the alternate compression and expansion of the coal mass on the conveyor, pools of water remained on the upper layers of coal, the added slimes often acting as a cementing agent for the coarser particles of coal and forming a relatively impermeable layer. The capital cost of the drainage conveyor was also high, and, in 1914, the successful designing of a simpler appliance for dewatering led to its abandonment.

Dewatering Screens.—The new method of dewatering consisted in passing the fine washed coal over fixed and shaking screens before delivery on to a scraper conveyor for loading into the storage bunkers. Fig. 218 is a diagram of the dewatering apparatus and Fig. 219 a view of the appliance in operation. In Fig. 42, page 159, a plan is given of the dewatering apparatus, which is made up of four sets of fixed screens, 17, and shaking screens, 18. The slurry from the settling tanks, 21, passes from the base of the tank through a pipe to a separate set of four fixed screens, from which the coal is discharged on to the shaking screen, 18.

In Fig. 218 the fine washed coal passes from the rewash-box on to the shoot, A, and to the fixed sieve, B, which is constructed of wedge wire. The wedge wires consist of rods of triangular cross-section. The floor of the fixed wedge wire screen is composed of a series of such rods fastened together to give a flat surface with transverse slits, through which the water, and only the finest particles of coal, can pass. Each of the spaces between the wires increases in width on the under side of the sieve, and this prevents the screen from being easily blocked by fine coal particles. The water passing through the fixed sieve, B, carrying with it some of the finest particles of coal, passes along the launder, C, and runs back to the settling tank. The coal, freed from much of the entangled water, passes on to the shaking sieve, D_1, constructed also of wedge wire. It is shaken along this screen into a second shaking screen, D_2, from which the coal passes into a launder, E, on to a conveyor for loading into the bunkers. The concentrated slurry from the settling tank passes from the launder, F, to the fixed sieve, G, where most of the water is removed, and passes into a launder, K, and thence to the settling tank along the lower launder, C. The fixed slurry sieve, G, has a greater inclination than the fixed fine-coal sieve, B, to allow the slurry to flow readily down the launder, H, on to the top of the fine coal on the shaking screen, D_1. The partly-dewatered slurry and fine coal pass together from D_1, to the second shaking screen, D_2, and thence through the launder, E, to the conveyor. The shaking screens are actuated by an eccentric (shown on the right-hand side of Fig. 218), which gives them a rapid to and fro motion.

This method of dewatering proved to be more efficient than the

older method of using a drainage band, and, since 1914, many of the older drainage conveyors have been removed. The change is simplified by the fact that the fixed and shaking screens occupy no more space than the old drainage band, and do not interfere with the general arrangement of the washery.

In 1926, an improvement was effected by Simon-Carves in treating the slurry on a separate set of shaking screens. By treating the slurry and the fine coal separately it is found that the dewatering is more efficient. For example, slurry may be dewatered so that only 26 per cent. of water remains, whereas, in previous practice the amount retained in the slurry would be of the order of 32 per cent. or more. At one South Yorkshire washery, the small coal, after dewatering on a drainage band, contained 14·0 per cent. of water, on an average for three months' working. When the drainage band was replaced by a set of shaking screens, the average moisture content of the coal was reduced to 11·1 per cent.

The Sherwood Hunter dewatering screens are similar to those already described. Although no provision is made for dewatering the slurry on a separate set of shaking screens, the efficiency of dewatering may be illustrated by the results obtained with the installation at Frickley Colliery, where the fine coal at the delivery end of the screen now contains, on the average, 18 per cent. of water instead of 23 per cent., the average moisture content when using the old drainage band.

Dewatering screens are now widely employed by nearly every washery constructor. They differ somewhat in details but all are essentially the same in principle. On the Continent it is common practice to dewater slurry on such screens (or " Zimmers ") and to use a spray of water near the feed end. This removes clay particles and enables the water content to be further reduced. One such dewatering screen is shown in Fig. 220.

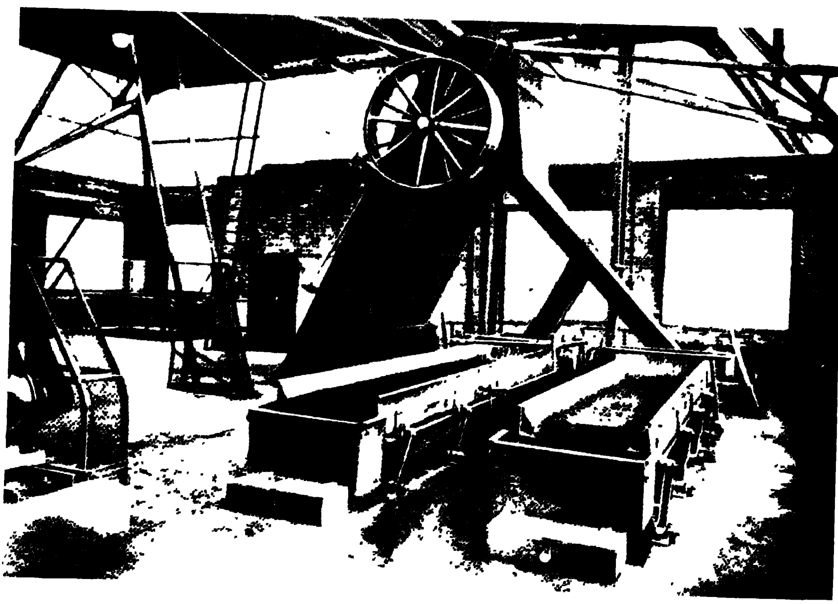

FIG. 220.—View of Zimmer Dewatering Screen Aufbereitung A.G.

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CHAPTER XXVI

DEWATERING: FILTERS, CENTRIFUGES AND DRYERS

In addition to the devices discussed in Chapter XXV, many other mechanical means have been successfully employed for dewatering washed coal.

These may be grouped into three classes: One, using suction (the Oliver filter); a second, using pressure (centrifugal dryers and filter presses); and the third, using direct heating (the Ruggles-Coles dryer). The intensity of the force exerted for dewatering increases in the order given: in the Oliver filter the suction used cannot be greater than 1 atmos. (30 in. of mercury); in centrifugal dryers, the force used may be equivalent to several hundred atmospheres; and, in direct-heat dryers, the mechanical equivalent of the energy used as heat is of a very high order. For example, in dewatering 10 tons of coal per hour by direct heat and removing 5 per cent. of water, the mechanical equivalent of the energy used is 1,250,000 ft. lb. per hour, or the equivalent of 3,230 h.p. It must not be thought that the cost of power for treating coal by these three methods is proportional to the figures given, which are only calculated to compare the intensities of the forces used. The greater the intensity of the force, the more completely can dewatering be effected, but the actual power consumptions are more affected by the mechanical difficulties of the application of the force. Thus in a suction filter, in order that a high vacuum can be attained, a thick bed of material has to be carried to prevent undue passage of air through the bed. Although the actual power consumed in creating the suction is small, the motion necessary to make the process continuous leads to a high gross power consumption. In centrifugal dryers, the power consumption depends largely on the thickness of the coal layer on the screens. With the use of a simple scraping device, however, the thickness of the coal layer can be considerably reduced. With a well-designed centrifuge the power consumption is low, the high intensity of centrifugal force is economically used, and large throughputs may be attained. In direct-heat dryers the energy used for coal drying may be "waste heat," and, in this event, power would only be required to rotate the dryer (which is usually cylindrical) and in exhausting the steam and waste gases.

SUCTION AND PRESSURE FILTERS

For the purpose of description, pressure filters (filter presses) and suction filters may be considered under one heading. Both

types have been used for dewatering froth-flotation concentrates, especially on the Continent, where froth-flotation processes are more generally used than in Great Britain. When considerable difficulty is experienced in reducing the moisture content of the washed coal to a reasonable figure, suction and pressure filters may be adopted, but they are relatively expensive. For froth-flotation concentrates, the expense may be justified, because less drastic methods are ineffective, and the removal of water increases the utility of the cleaned product.

The Oliver Filter.— The Oliver filter consists of a drum 8 ft. long and 8 ft. in diameter, on the outside of which a 36 mesh brass wire cloth is wrapped and held in position by wire wound spirally. The drum is supported on trunnions, and the lower half is submerged

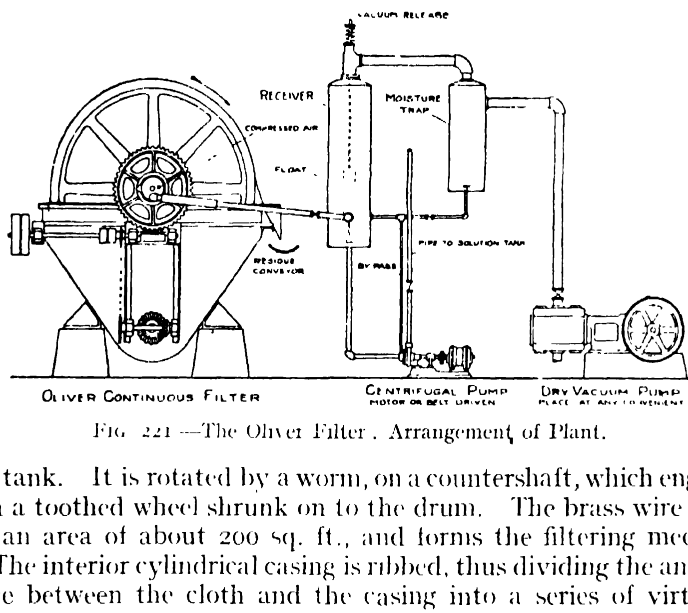

FIG. 221.—The Oliver Filter. Arrangement of Plant.

in a tank. It is rotated by a worm, on a countershaft, which engages with a toothed wheel shrunk on to the drum. The brass wire cloth has an area of about 200 sq. ft., and forms the filtering medium.

The interior cylindrical casing is ribbed, thus dividing the annular space between the cloth and the casing into a series of virtually independent shallow compartments, each 1 in. deep. Each compartment is connected by a pipe to a valve placed at the centre of the cylinder to which suction can be applied.

The froth-flotation concentrate, containing about equal proportions of solid and water, is introduced into the tank in which the drum rotates, and is maintained at a constant height, homogeneity being assured by mechanical agitation. As the drum rotates, and suction is applied to the inside of the filter cloth, a cake is built up on the outside of the cloth. During the passage of the screen through the tank, the cake collected on the cloth is concentrated, compared with the free material in the tank, and, when the screen emerges above the water level, true dewatering commences. The

water drawn off through the filter cloth passes along the radial pipes in the interior of the drum. The vacuum pump creating suction is connected with the radial pipes in the interior of the drum through an automatic valve. A reservoir and water-trap tank are interposed between the automatic valve and the vacuum pump. The water collected from the filtered material is removed from the system by a centrifugal pump (Fig. 221).

When the filter has almost completed one revolution, the filter cake is removed by the application of compressed air to its underside. This change-over, from the vacuum to compressed air, is made through the automatic vacuum valve which, at a suitable moment, cuts off the connection of the receiver to the vacuum pipes and connects them to a compressed-air supply. The cake is loosened from the filtering cloth so that a fixed scraper readily removes it on to a belt conveyor. The screen meshes are thus cleared and a clean filter surface re-enters the tank for another cycle of operations.

At Oughterside (Scoular and Dunglinson, *Iron and Coal Tr. Rev.*, 104, 1112), from 26 to 28 tons of material, containing 50 per cent. of water, were handled per hour in an Oliver filter, and the water content of the dewatered coal was 17 per cent. on the average. The power requirements were 45 h p. to drive the filter, vacuum pump and a centrifugal water pump. Tests made with an intermittent centrifuge with charges of 15 cwt., reduced the water content to 14 per cent. O. Schäfer (*Stahl und Eisen*, 1) records a figure of 20 per cent. for the moisture content of slurry after cleaning by froth flotation and dewatering in a filter of the Oliver type. In the German plant the cleaned slurry from the froth-flotation plant was " thickened " in a cone before delivering to the filter.

A battery of twenty Wolf filters is in operation at the Dutch State Mines, Limburg, treating fine coal (slurry) which has been cleaned by the Kleinbentinck froth-flotation process. From this material a dried product containing 20 per cent. of water was produced at the rate of approximately 1·35 tons per 10 sq. ft. of filter surface per hour, using a layer up to ½ in. thick, at 5 r.p.m. At Mont Cenis the concentrates from the Ekof froth-flotation plant are led to rotary filters containing 69·4 per cent. of water, which is reduced to 20 per cent. by the suction-filter treatment. One filter had a filtering surface of 10 sq. m (107·5 sq. ft.), and produced 840 kg. of dried material per square metre per hour (8¼ tons of material containing 20 per cent. water per hour). The thickness of the layer on the cloth was about ¾ in. The speed of rotation was 1·1 r.p.m. The vacuum applied was 420 mm. of mercury. Jute sacking was used underneath the outer covering of bronze-gauze to regulate the filtering action.

Grounds (*Fuel Economist*, 560) also records the use of rotary filters to filter the waste water from lignite and brown coal washeries and briquetting factories. By treating per square metre of filtering surface, 8,000 litres (1,760 gall.) per hour of liquor con-

taining only 1·45 per cent. of solid matter, a clean filtrate was produced. The filter cake (10 mm. thick) contained 37·3 per cent. of water. Cheap cotton cloth was used as the filtering medium.

Many makers supply rotary filters similar in principle to the one described. Such filters are the Portland, the Wolf, the Zenith and the American. The one illustrated in Fig. 222 is a Wolf filter.

The Dorrco Filter.—A rotary vacuum filter of quite different design has recently been marketed by the Dorr Company, New York. The filter is cylindrical, as is the Oliver, but instead of making the cake on the outside of the filter drum it is produced on the inside perimeter. The advantage of this arrangement is that the vacuum

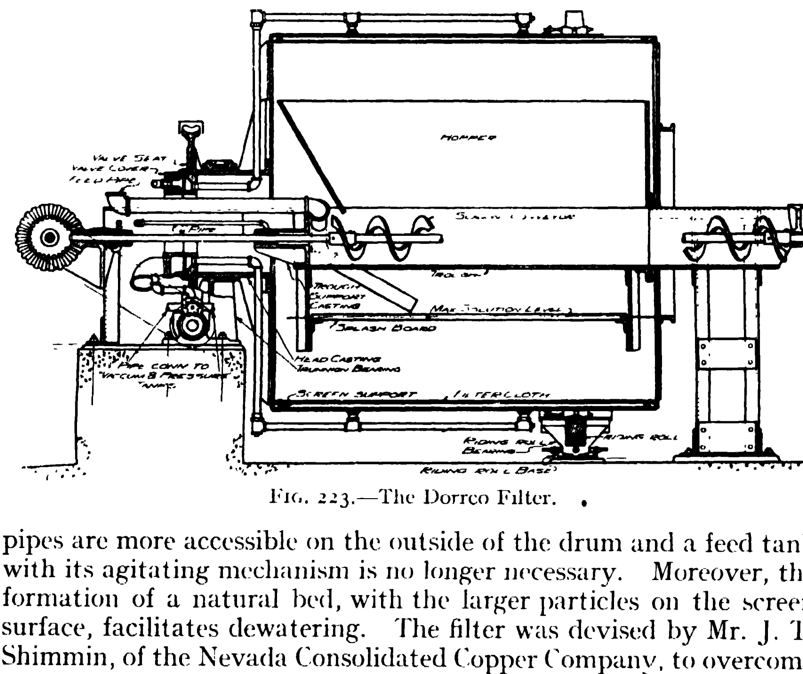

FIG. 223.—The Dorrco Filter.

pipes are more accessible on the outside of the drum and a feed tank with its agitating mechanism is no longer necessary. Moreover, the formation of a natural bed, with the larger particles on the screen surface, facilitates dewatering. The filter was devised by Mr. J. T. Shimmin, of the Nevada Consolidated Copper Company, to overcome difficulties which were experienced when using the ordinary type of rotary vacuum filter. With metalliferous ores the larger and heavier particles tended to settle out in the feed tank despite constant agitation, and the drums had frequently to be raised in order to dig out the settled ore particles. The Dorrco filter illustrated in Fig. 223 is supported on trunnion bearings at the feed end and by a tyre bearing on rollers at the other end. The drum is rotated by means of a worm and a worm-gear drive. The filtering cloth on the inside of the drum is built up in sections which may be replaced individually. The material to be dewatered is fed to the drum through a feed pipe which discharges on to a splash-board in the lower half of the drum. A natural bed is formed on the filter surface, and suction is applied

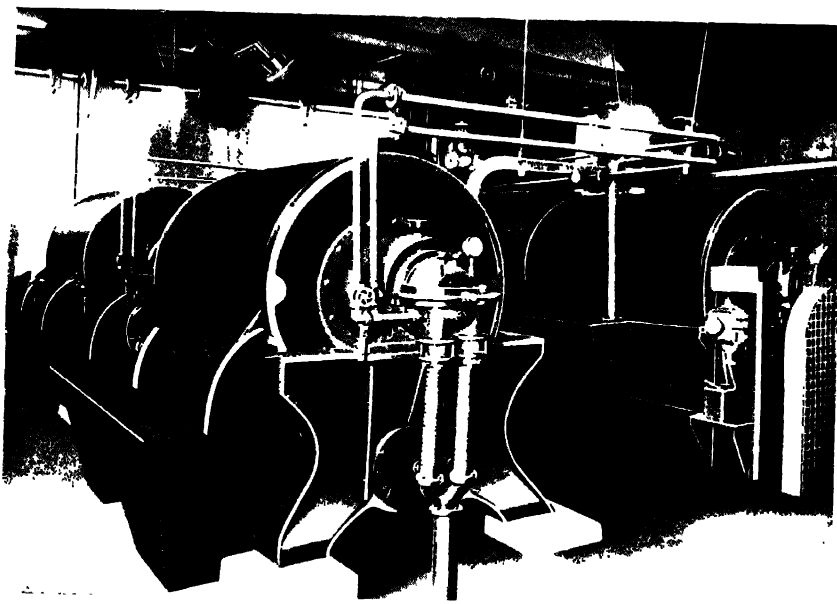

FIG. 222.—View of Rotary Filter (Wolf).

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to each panel of the screen through an automatic valve. The cake formed on the inside of the drum is carried round to a suitable point where the filter screen is caused to vibrate to loosen the cake and enable it to drop into an internal hopper. From the hopper the dried material passes to a trough in the centre of the drum, whence a screw conveyor discharges it.

With the new filter, metalliferous ore slimes through 65 mesh size were dewatered from 34·5 to 12·0 per cent. at the rate of about 5 tons per hour in a filter of 13 ft. 1½ in. diameter. The power requirements were 6 h.p. for the filter drive and 13½ h.p. for the vacuum pump. With a mixture of flotation and table concentrates, in which 70 per cent. of the feed was fine enough to pass through 65 mesh, the average capacity of the filter was increased to 10·2 tons per hour in dewatering from 40·0 to 11·0 per cent., and using only 10 h.p. Finally, the capacity of the filter was raised until it treated over 20 tons of concentrates per hour.

The Plan Filter.—The Plan filter is used at the Dutch State Mines, Limburg, for material which, owing to difficulties in keeping large particles in suspension in the supply tank, and in making a cake, is too large to be treated successfully by a rotary filter. It is also used in Germany for dewatering slurry. The filter consists of a horizontal table built up of perforated metal sheet overlaid with fine wedge-wire panels. The underside of the table is connected by a number of suction pipes to a central discharge pipe in the circuit of a vacuum pump. The table is rotated by suitable mechanism. The filters used at Limburg are circular, the system of suction pipes and rotating mechanism forming an elaborate understructure.

The Plan filter illustrated in Fig. 224 is manufactured by Fr. Gröppel. The central portion of the table is cut out and the suction pipes are arranged radially. The understructure is therefore simplified and less head room is required. The material to be dewatered is fed to the table through the pipe under the elevator and, when the table has completed one revolution, the dewatered material is removed by means of the elevator.

At Mont Cenis, Westphalia, a Plan filter of 1·5 sq. m. (16 sq. ft.) filtering surface is used to treat a slurry which contains only 26·6 per cent. of solids. The moisture content of the product is 25 per cent. The slurry is built up to a thickness of about ⅕ in., and the filter revolves slowly at a speed of 24 revolutions per hour, a suction equivalent to 6 to 7½ lb. per sq. in. being applied. The filtrate is only slightly cloudy.

Filter Press.—O. Schäfer (*loc. cit.*) records that use was made of a filter press at Zwickau, Saxony, to dewater the concentrate from a froth-flotation plant, cleaning slurry. Each plant consisted of forty-two chambers of 1 sq. m. filter surface. Six presses, with seventy-five operations per twenty-four hours, would give an output

of 300 tons of slurry containing 20 per cent. of water, or 50 tons of such material, per press, per twenty-four hours. The arrangement at Zwickau is shown in Fig. 225. Thickened slurry from the bottom valves of a number of settling tanks could be run through a launder into a tank. From this the lower slurry container could be filled, and, on closing the valve between the two vessels and applying compressed air to the lower container, the slurry was fed into a press. After dewatering, the press was opened and the dewatered

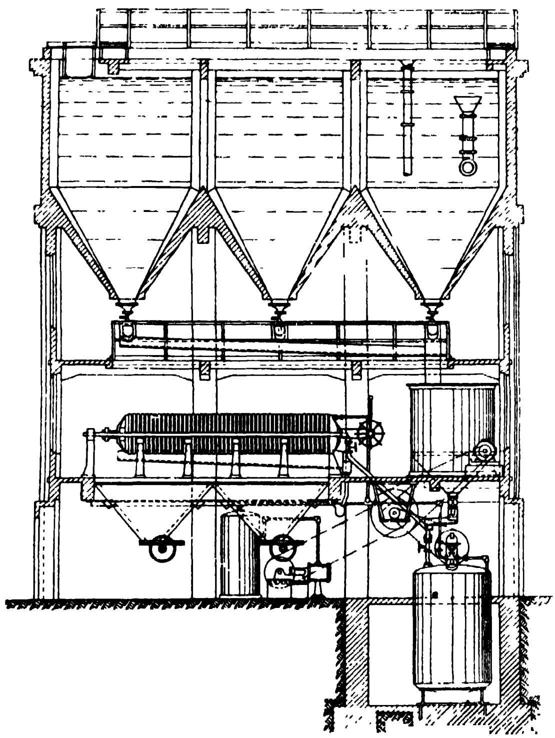

FIG. 225.—Filter Press suitable for Coal Dewatering.

slurry fell into the loading hoppers below. The details of the press are not recorded.

Details of a press used in chemical works are shown in Fig. 226. The press consists of a number of perforated plates which slide along bars on either side of the press during the opening and closing. A section through a number of plates is also shown; each chamber, enclosed by two plates, is in communication with channels *i*, *k*, and *m*, by cross-channels. The particular press illustrated is of the "total extraction" type, and allows a pressed cake to be washed with water admitted through the channel, *m*, and passing out through the channel, *k*. This operation is unnecessary in coal dewatering,

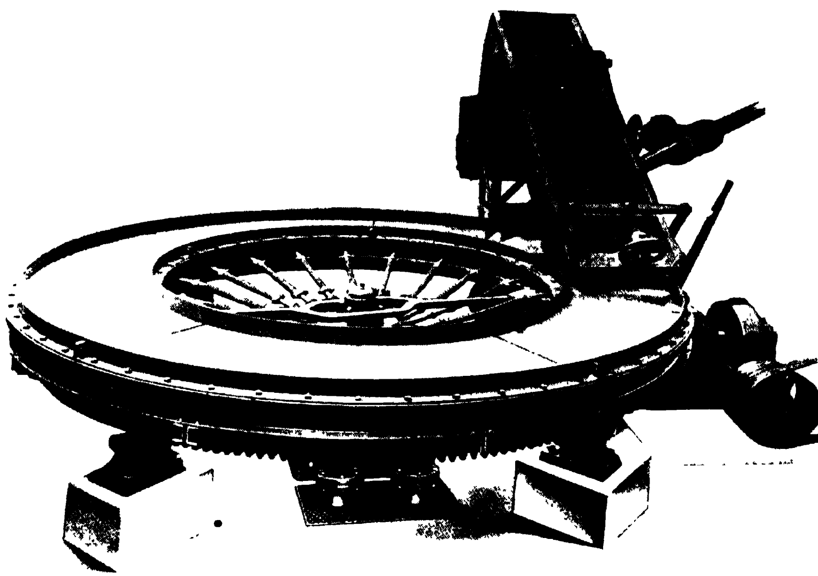

Fig. 224. The Plan Filter.

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and a single channel only would be required. Suitable filter cloths are used on the press plates, which are then closed and the chambers filled with the material to be treated. The press is tightened by means of a screw, or pneumatically as in Fig. 225. When the maximum pressure has been applied, the press is opened and the pressed cakes drop out of the chambers.

At Zwickau the cleaned slurry is dewatered to 18 per cent. water content. The power requirements are 7·5 h.p. for a throughput of 10 tons of wet slurry (containing 50 per cent. of water) or 5 tons of dry slurry per hour.

CENTRIFUGAL DRYERS

A number of continuous centrifugal dryers have been successfully used for dewatering washed coal. With some of them large throughputs have been obtained, and it has been found possible to reduce the moisture content of coal from 15 or 20 per cent. to less than

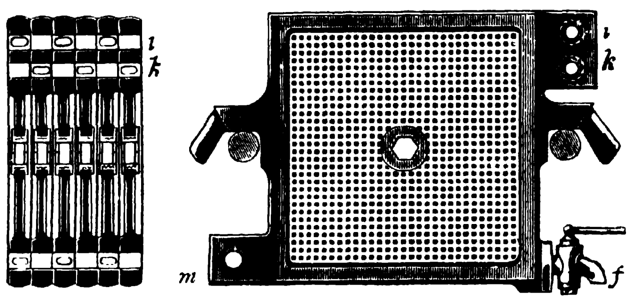

FIG. 226 —Details of Filter Press.

10 per cent. The ground space required is small and the power consumption not excessive. The chief disadvantage, however, is the liability to breakdowns and the cost of repairs, especially of renewing the screens. The mechanism is often somewhat complicated, and special labour is therefore necessary when breakdowns occur.

The Elmore Centrifuge.—The Elmore centrifuge was originally designed to remove liquor, acid, or water from crystals in chemical works, and to replace the intermittent type of centrifugal dryer usually used. It was argued that the weight of a heavy intermittent charge greatly reduced the speed at which the centrifuge ran, and, as the centrifugal force exerted varies as the square of the velocity, the force applied to the material was considerably reduced. In continuous operation, a very small charge of material was present in the centrifuge at any one instant, and it was possible to increase the speed of rotation without increasing the strain on the centrifuge.

The higher centrifugal force developed ensured more complete removal of adherent liquor.

The general design of the Elmore centrifuge is shown in Fig 227, It consists of a rotating basket, or sieve, with a scraper rotating at a somewhat lower speed inside it to remove the dried material. The

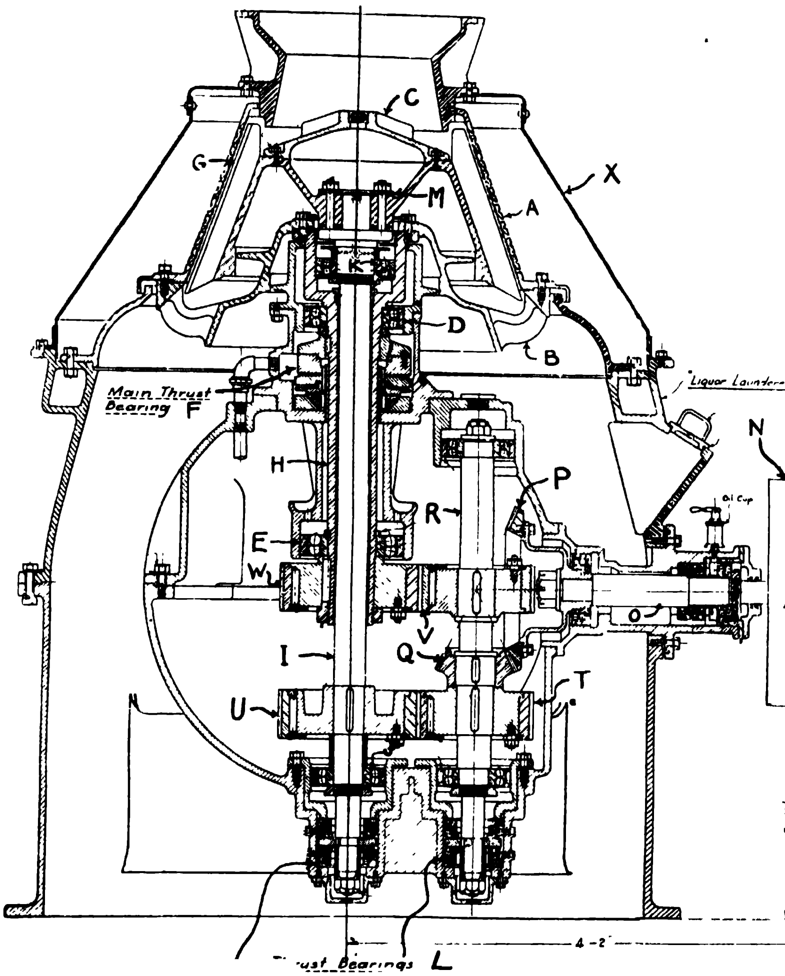

FIG. 227.—The Elmore Centrifuge

basket, A, is made of phosphor-bronze, monel metal or steel (according to the nature of the liquid to be removed), and is cast in one piece Inside this basket a removable screen is fixed as the actual filtering medium. The basket rests on the rim of the cast-steel spider, B, carried by the hollow or quill shaft, H, which runs in the ball bearings, D and E, housed in cages. The thrust is taken by the bearing, F The renewable scraping flights, G, are attached to

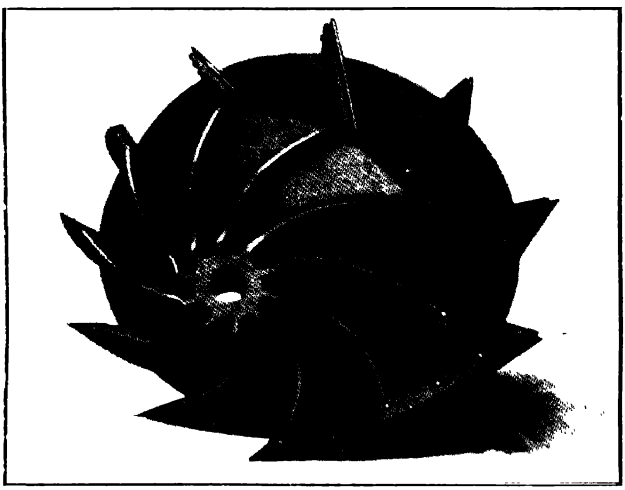

FIG. 228.—Distribution Cone and Scrapers of Elmore Centrifuge.

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a distributing cone to which the distributing paddles, C, are also fixed. The distributing cone is carried by the internal solid shaft, I, running in the ball bearings, J, and K, in cages, which are kept in vertical alignment by suitable guides. The thrust is taken by the bearing, L, at the bottom of the shaft. The distance between the scraping flights and the screen may be varied by adjustment of the "shims," M, set above the flange of the solid shaft, which raises or lowers the distributing cone carrying the scraping flights.

The two shafts are driven by differential bevel gearing. The driving pulley, N, attached to the shaft, O, drives the bevel gear, P, which actuates the forged steel bevel pinion, Q, on the countershaft, R. This countershaft also carries the differential spiral gears, V and T. The gear, T, drives the spiral gear, U, keyed to the internal solid shaft which drives the scrapers and distributing cone. The gear, V, drives the spiral gear, W, keyed to the hollow shaft which drives the basket and screens The gear wheels are enclosed in an oil-tight gear-case which is filled with oil to the level of the bottom face of the spiral gear, U. A centrifugal oil pump circulates oil to the ball bearings, the various gear wheels and the stationary countershaft. The difference in the speed of the two rotating shafts is made one revolution for every 100 revolutions of the basket shaft.

For crystal drying, the centrifuge is made in four sizes—namely, 10, 24, 36 and 48 in. diameter between the bolt holes of the lower end of the basket; the speeds of rotation are 3,000, 2,100, 1,600, and 1,100 r.p.m. respectively. For coal drying, the 48 in. centrifuge is used at a basket speed of 550 r.p.m. Coal is admitted to the hopper at the top and is distributed by the paddles, C, on the distributing cone (see Fig. 228, which shows the paddles and the scraping flights on the distributing cone). The coal is then flung on to the screen fixed inside the basket, G, and is removed by the scraping flights and falls through the annular space between the gear-case and the outer casing on to a conveyor. The water forced through the interstices of the basket impinges on the shield, X, and is collected in a launder.

The 48 in. centrifuge adopted for coal drying has a capacity of 80 tons of wet coal per hour, and dewaters it to approximately 7 per cent. Four dryers, each with a capacity of 80 tons per hour, have been installed at the Middle Fork washery, Benton, Ill., and also at No. 8 mine of the Tennessee Coal, Iron and R.R. Co., Birmingham, Ala. The power requirements are 30 h.p., and the floor space 7 ft. by 7 ft. 6 in. per unit. At the coking plant of the Woodward Iron Co., Bessemer, Ala., coking coal through $\frac{5}{8}$ in. mesh is dried. One set of screens is said to last for the time necessary to dry from 20,000 to 30,000 tons of coal. At the Emma pit of the Dutch State mines, three 36 in. centrifuges have each a capacity of 30 tons per hour, using 40 h.p. each. At Parkgate, Yorkshire, one centrifuge has a rated capacity of 75 tons per hour, but has been overloaded considerably above this figure. The power consumption is about

$\frac{1}{2}$ h.p. per ton. The average result of eighteen daily tests on this centrifuge shows that the moisture content of the coal was reduced from 19·8 per cent. to 7·4 per cent.; the lowest water content of the treated coal was 6·4 per cent. The breakage caused in passing through the centrifuge is shown by the following average figures of three daily tests :—

Size (in.).	Before Treatment.	After Treatment.
$\frac{1}{2}$–$\frac{1}{4}$	51·1	17·5
$\frac{1}{4}$–$\frac{1}{8}$	24·3	39·6
$< \frac{1}{8}$	24·2	42·7

The water removed from the coal at Parkgate, including the fine coal passing through the screen, was lifted by an ejector to the top of the slurry-settling tank and was therefore retreated. This reduces the effective throughput of the centrifuge, but avoids the accumulation of slurry.

For coal dewatering, the outer steel basket has peripheral slots about $\frac{1}{4}$ in. wide. Inside this basket is fixed a screen which may be made either of thin phosphor-bronze sheet with inclined slots, $\frac{1}{2}$ in. long and $\frac{1}{16}$ in. wide, or of punched steel plate with $\frac{1}{16}$ in. round holes. The removable screens are only thin, but some difficulty is experienced in laying them truly against the basket. Any irregularity of the surface tends to be worn by the scraping flights, so that holes are formed. A torn edge causes coal to build up against it, and may throw sufficient back pressure on the machine to cause a stoppage. This is the weakest feature of the centrifuge, and may cause serious inconvenience in operation.

The large throughputs obtained suggest that the coal is not allowed to remain long on the screen, and that the layer of coal carried is comparatively thin. The thickness is fixed by the distance of the scraping flights from the screen. The low power consumption is, no doubt, partly due to the thinness of the coal layer carried, and partly to the conical shape of the basket which deflects the coal downwards after impact, so that only a small duty is imposed on the scraping flights to aid the downward movement of the coal. In the Hoyle centrifuge, the coal after impact is not helped to move downwards by the screen (which is vertical), and all the duty of moving the coal through the dryer is imposed on the scraping device. In the Carpenter centrifuge the inclination is large enough to make a scraping device unnecessary. The power consumption for a given coal throughput is lower for the Elmore dryer than for other continuous centrifuges.

The Hoyle Centrifuge.—There are two Hoyle continuous centrifuges, one designed as a slurry separator or thickener, and the other more particularly for coal dewatering. The slurry separator was designed for the removal of suspended solids from water rather than for the removal of water from solids. It consists of a solid

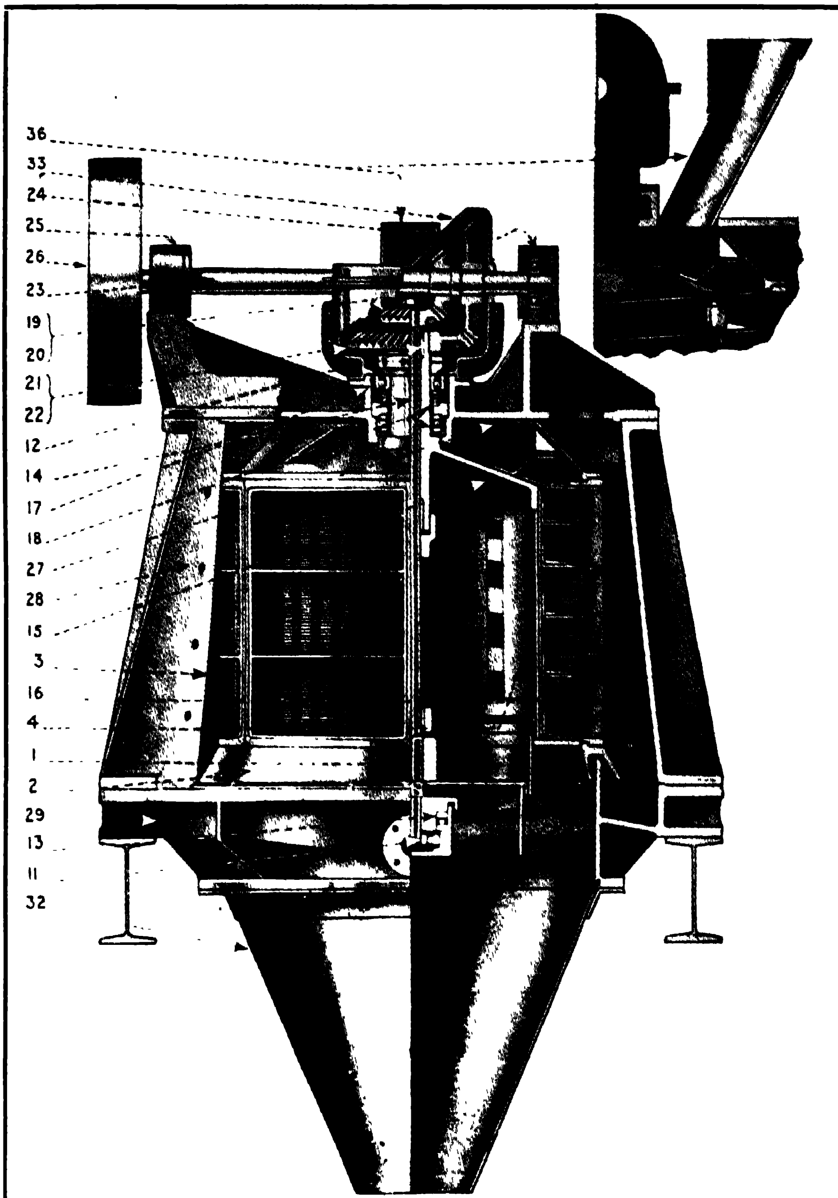

FIG. 229 - The Heyle Centrifuge. First Design.

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conical vessel mounted on a hollow shaft and driven at a speed of 800 to 2,000 r.p.m. A second hollow shaft carries a spiral and is rotated in the same direction as the outer vessel, but at a slightly lower speed, by differential gears. The slurry water is fed into the upper hollow shaft and flung against the conical vessel. The concentrated slurry is scraped from the conical member by the spiral and carried upwards, being discharged into the outer casing. Here a spiral prevents the slurry from adhering to the casing and causes it to fall down on to a conveyor. The clarified water is discharged through suitable openings.

The Hoyle centrifuge is built on the same general principles as the Elmore centrifuge, in that a screen basket and scraper spiral are rotated by differential gearing. The earlier (1922) design is illustrated in Fig. 229. A central mild steel shaft, 1, has a cast-steel spider, 2, which carries a screen, 3, built in sections of brass wedge wire. The weight of these parts is taken by the thrust ball bearings, 13, and the shaft revolves in the lower ball bearings, 11, and the upper gun-metal bush, 12. The spiral scroll, 16, is fitted to steel plates and fan blades, 4, fastened to a distributing cone, 15, attached to the hollow cast-steel shaft, 14. The weight of these parts is taken by the thrust bearing, 18, and the hollow shaft rotates in the ball bearings, 17.

The countershaft, 23, actuates gears, 19, 20, and 21, 22, to drive the inner solid and the outer hollow shafts. The gears are made of special steel with spiral teeth, and are housed in an oil bath in a gear case, 33. The lower ball bearings, 11, and 13, are lubricated from grease cups. The outer casing, 28, has two removable sections for inspection, and to allow renewal of the screen sections.

Coal is admitted by the shoot, 36, and is thrown by the distributing cone on to the screen from which it is scraped by the spiral scroll. The scroll increases in pitch towards the bottom, so that a thinner layer of coal is produced on the lower levels of the screen, and dewatering is thereby aided. A current of air is induced by the fan blades, 4, and is forced through the coal, leaving the outer casing through holes in the cover, 27. The dewatered coal drops into the lower coned section of the dryer on to a conveyor. The water removed, together with some of the finest particles, is collected in a launder in the outer casing and is run into a sump. In continued operation, the distance between the individual wires of the screen increases through the abrasive action of the coal, and larger quantities of coal pass through until the difficulty of handling this material becomes so serious as to compel the renewal of the screens.

The machine illustrated operates at a speed of 600 r.p.m., and, with a diameter of 4 ft., would give a centrifugal force 240 times the force of gravity. The scroll is composed of four "threads."

This type of centrifuge was first erected at Tinsley Park Colliery, in 1922, and dewatered the coal from the drainage hoppers to a moisture content of 9 to 10 per cent. About 30 tons of coal were

handled per hour with a power consumption of about 1 h.p. per ton. The renewal cost of the screens proved to be somewhat expensive, and the design was adapted to attempt to overcome this difficulty.

In the newer design the chief feature was the use of steel wedge wire in place of the brass wedge wire previously used. The height of the screen section was also considerably reduced, as will be seen from Fig. 230. The mild steel spiral was fitted with case-hardened steel renewable strips. The spiral was mounted on a cast-steel drum, and the gun-metal bush previously used at the upper end of the hollow shaft was replaced by a ball bearing. The gears were made of nickel-chrome steel, and the shafts of forged steel. A centrifugal oil pump was used to lubricate the driving gears.

Even in the newer design, the wear of the screens proved to be considerable, for, as previously stated, the vertical alignment of the screen imposes a heavy duty on the scrapers in moving the coal downwards, resulting in an unavoidable wearing of the screens. The wedge-wire screens themselves are more costly to replace than the punched steel plates of the Elmore and the Carpenter centrifuges. Coal is not scraped off the screens so quickly by a spiral scraper as by the series of vertical scrapers used in the Elmore dryer, so that a lower throughput is obtained. The thicker coal layer and the heavier duty on the scrapers increases the power required for a given throughput.

The Carpenter Centrifuge.—The Carpenter centrifuge was invented by H. B. Carpenter, the superintendent of the by-product plant of the Colorado Fuel and Iron Company, Pueblo, Colo., U.S.A., where six units have been in operation for four years. The patent rights in Great Britain, the British Empire (except Canada) and Continental Europe, have been acquired by the Woodall-Duckham Co., Ltd.

The centrifuge is illustrated in Fig. 231. It differs considerably in design from the Elmore and the Hoyle centrifuges, in that no subsidiary scraping device is used and complicated differential gear is not required. It consists simply of a stepped, truncated cone fastened to a vertical shaft by spider castings at the top and bottom. This arrangement is shown in more detail in Fig. 232. Each step of the cone comprises one screen. The weight is taken by a ball bearing at the top (Fig. 231) and the shaft rotates in upper and lower roller bearings, the upper bearing being housed with the thrust bearing in a suitable cage. These roller bearings also take the side thrust, and, with the ball bearings, are lubricated by grease from grease guns. The shaft is driven from a pulley on a countershaft through two bevel pinions in a gear case, lubricated by an oil pump. In the latest design, the larger machines are driven by a direct-coupled motor through a friction-coupling to the bevel-gear shaft.

The wet coal is fed into the feed hopper, A, by a constant feeding device on to a distributing disc, B, whose distance from the

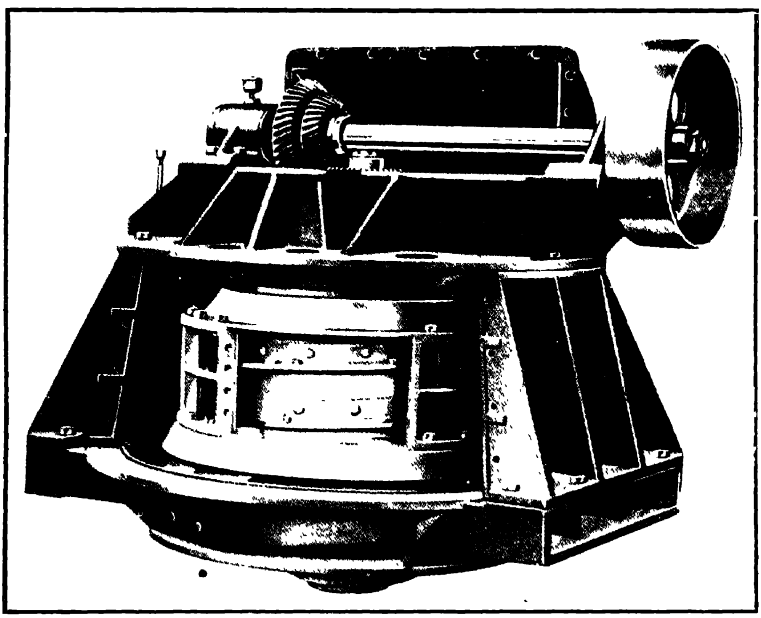

FIG. 230. The Hoyle Centrifuge—Later Design.

cone may be adjusted by the set screws seen below the disc. The coal is flung from this disc on to the first screen, C, from which it works its way down to the first step, E, Fig. 231 (seen more clearly

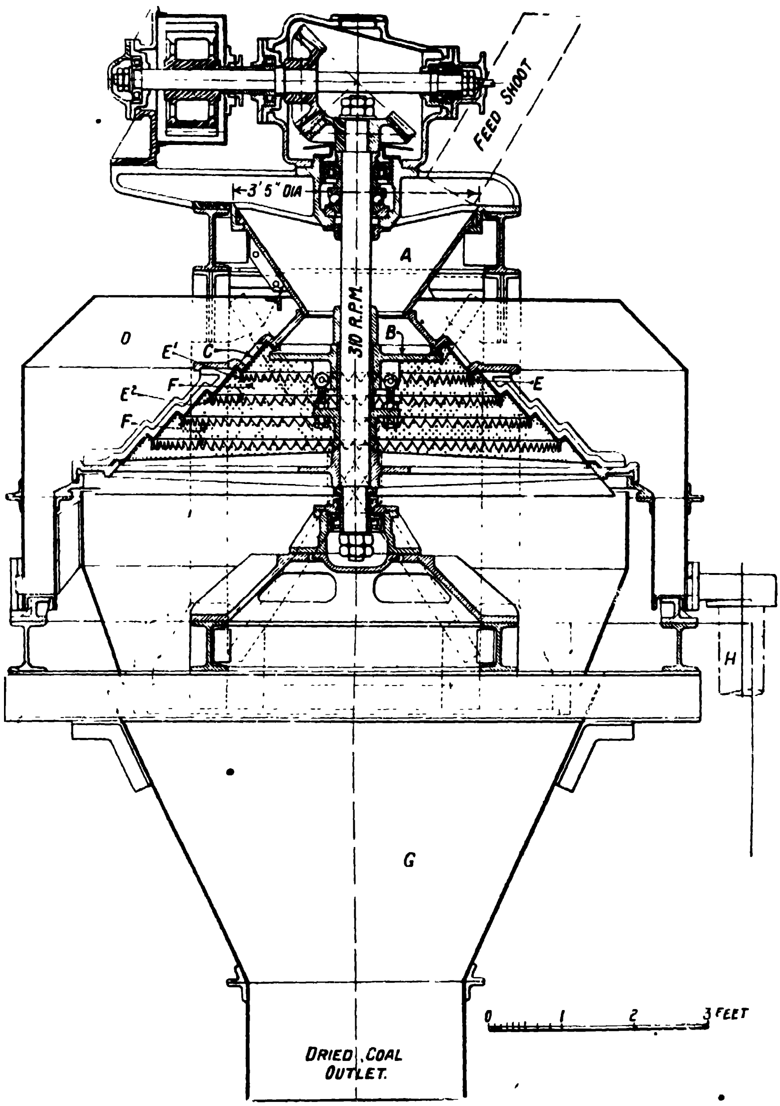

FIG. 231.—The Carpenter Centrifuge.

in Fig. 232). Here a set of serrated teeth breaks up the coal mass, after which it is flung on to the second screen, E[1]. The coal mass is again broken up at the successive steps by the serrated teeth, F, and is successively treated on the screens, E[2], etc., before it

drops into the delivery hopper, G (Fig. 231). The coal is subjected to centrifugal force, which increases with increasing diameter of the centrifuge. As the water content of the coal is reduced, the coal is subjected to a greater centrifugal force. This arrangement confers to the Carpenter machine an advantage (shared by the Elmore centrifuge to some extent) over the Hoyle dryer (in which the centrifugal force is at a maximum on the first impingement of the coal on the screen) for the gradual increase in the centrifugal force used has the desirable result of reducing the local wear of the screens. The water passing through the screens is collected in a launder and removed through two outlets, H (Fig. 231).

The screens are made of $\frac{1}{8}$-in. steel plate with $\frac{1}{8}$-in. holes, spaced $\frac{3}{16}$ in. between centres, and are renewable in sections. In the centrifuge installed at Nunnery Colliery, Sheffield (T. A. Long, *Gas World*, 1926, 84 (Coking Section, June), 17), about 5 per cent. of fine coal (slurry) passes through the screens with the effluent water. If this slurry were allowed to run directly into settling ponds, over 15 tons would be produced per day, and the handling and disposal in bulk of this quantity would prove expensive ; moreover, it would prove wasteful, for such material contains over 30 per cent. of water and aggregates in masses, which, if charged to the coke ovens, causes " stickers." The effluent is therefore pumped to a dewatering screen of $\frac{1}{16}$ in. mesh at the top of the storage bunker, and the dewatered solids are added to the washed coal for retreatment in the centrifuge. The effluent water, containing only a small amount of solids in suspension, is then run to settling troughs, and any further solids deposited are returned to the bunker once more. Although there is a constant rehandling of about 15 tons of material per day, this is the best way of dealing with it, for the slurry is then admixed as uniformly as possible with the undried coal, and, moreover, is subjected to a second treatment in the centrifuge before charging to the ovens.

When the machine was first installed at Nunnery it was run at a speed of 350 r.p.m., and reduced the moisture content of the coal (through $\frac{7}{8}$ in.) to about 5 per cent. At this speed the coal was broken up so much that it was unnecessary to use the crusher before charging the coal to the ovens. To reduce the breakage, and so to avoid undue wear of the machine, its speed was reduced to 270 r.p.m., at which speed the coal was dewatered from 18 or 20 per cent. to 6 or 7 per cent. Forty tons of coal were treated per hour, with a power consumption of under 1 h.p. per ton of coal treated. The breakage of the coal on centrifuging is shown by the following percentage figures :—

Size (in.)	Before Treatment	After Treatment.
$> \frac{1}{8}$	78·0	55·0
$\frac{1}{8}-\frac{1}{16}$	10·6	18·5
$< \frac{1}{16}$	11·4	26·0

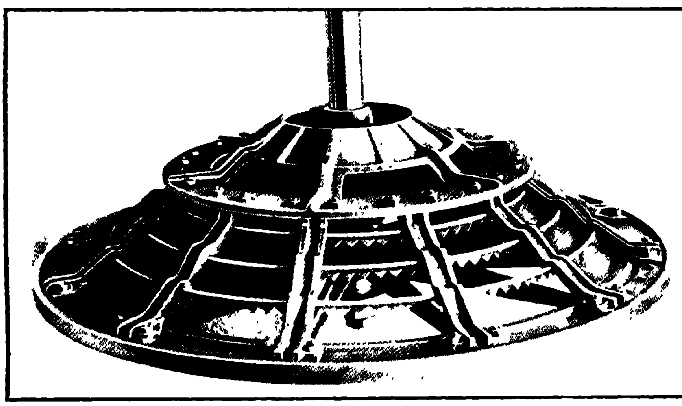

FIG. 232. Details of Carpenter Centrifuge.

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In a period of nine months, 70,000 tons of coal were treated in the centrifuge. Two shifts were worked per day, and the centrifuge was worked for about four hours of each shift. During this period, the top section of the screen was renewed twice, and the second and third sections once, the original bottom section being unchanged, The serrated teeth in each section were renewed once. The total cost of repairs amounted to 0·12*d*. per ton of coal treated.

At the Pueblo Works of the Colorado Fuel and Iron Company, the coal treated in the centrifuge contained an average of 90 per cent. of material through $\frac{1}{4}$ in., 60 per cent. through $\frac{1}{8}$ in., and 15 per cent. through $\frac{1}{32}$ in. A series of tests made over a period of seven months gave the results recorded in Table 133 (F. J. G. Duck, *Coal Age*, 1927, 31, 219).

TABLE 133.—RESULTS OF DEWATERING AT PUEBLO, COLO.

	Water per cent. in Coal.		Per cent. of Original Water Removed.
	Before Treatment.	After Treatment.	
Maximum dewatering . .	22·4	6·2	79·0
Minimum dewatering . .	16·8	4·7	70·0
Average dewatering . .	19·9	5·5	72·8

The centrifuges at Pueblo are said to have an average capacity of 100 tons per hour per unit, using 75 h.p. at a speed of 360 r.p.m.

The Carpenter centrifuge is the simplest of all the centrifuges described; moreover, the results obtained for dewatering are as good or even better than those reported for other centrifuges separating at much higher speeds, and of more complex design. The cost of renewals of those parts subjected to the greatest wear is probably smaller than for other centrifuges which have been tried in Great Britain. This is no doubt due to the use of simple steel plates, but also to the absence of a scraping device, for a layer of coal is always left on the screens to protect them from undue wear. Nevertheless, the working results show that the outward passage of water is not prevented. The large angle of inclination to the vertical of the screen surface, by deflecting the coal in a downward direction after impact on the screen, is probably one of the chief factors that allow a scraping device to be eliminated. The impact on the screen surface is also less direct than in the Hoyle or Elmore centrifuges, thus reducing wearing of the screen plate.

Four Carpenter centrifuges are in operation in Great Britain, and are giving satisfactory results. As previously noted, the power consumption of the Carpenter centrifuge per ton of coal is

higher than in the Elmore centrifuge, in spite of the fact that only one drive is used. The absence of a scraping device produces thicker coal layers on the screens, which cause a higher consumption of power, but reduce the wear on the screens, which are protected by coal. This compensation, which reduces the liability to breakdowns, is probably well worth the extra power consumption involved.

The Simplex Centrifuge.—Another continuous centrifuge which has been used at the West Ardsley Colliery, near Wakefield, Yorkshire, is the Simplex, invented by Mr. Fabry, of the Simplex Coke Oven Co., Ltd., Sheffield. Unlike the other centrifuges described, the axis of the machine is disposed horizontally (Fig. 233). The horizontal shaft is connected by a spider wheel, A, to a truncated conical shell, to which a short cylindrical screen, G, is attached. The screen is composed of a series of circular bars held in position by grooved distance pieces bolted to circular angles, which are riveted to the conical shell. The wet coal is fed into the machine through a shoot into a feed worm, *c*, which is bolted to the spider wheel and revolves with the shaft in the worm casing, D. The outlet end of the worm casing is cut helically to distribute the wet coal uniformly over the inside of the conical shell. A circular plough rotates on an idle shaft which is bolted to the casing and (as shown in Fig. 233) is inclined at an angle of 4 deg. from the vertical. The plough, which has a clearance of about ¼ in. from the screen, moves the coal towards the outlet.

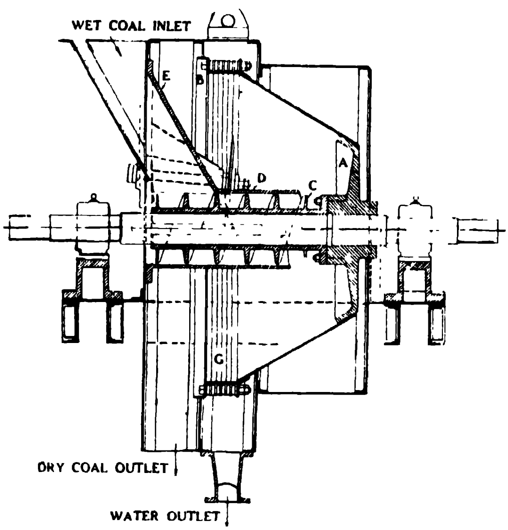

FIG. 233.—The Simplex Centrifuge

The coal from a trough washer is drained on a drainage conveyor and falls into the centrifuge hopper. The centrifuge, operating at 470 r.p.m., treats an average load of 20 tons of wet coal per hour, consuming 16 to 21 h.p. in dewatering the coal from 20 per cent. to 9·5 per cent. When the centrifuge was overloaded by 100 per cent., the same horse power was used, and wet coal was dewatered from 28·2 per cent. to 10·8 per cent. An average of four daily tests shows that coal of 15·2 per cent. initial water content was dewatered to 8·0 per cent. of water. The following figures show the breakage due to centrifuging:—

Size (in.).	Before Treatment.		After Treatment
$> \frac{1}{4}$	32·5	..	15·2
$\frac{1}{4}-\frac{1}{8}$	33·1	..	19·1
$\frac{1}{8}-\frac{1}{16}$	18·5	..	27·5
$\frac{1}{16}-\frac{1}{28}$	5·8	..	15·7
$< \frac{1}{28}$	10·1	..	22·5

The Wendell Centrifuge was invented by Mr. C. A. Wendell, of the Illinois Steel Company, and is made by the Link Belt Company. It is illustrated in Fig. 234. It consists of an inclined screen carried by a hollow shaft which is driven by bevel gearing from the top of the centrifuge. A solid shaft, inside the hollow one, carries two feed spouts which deliver the wet coal from a hopper on to the screen. Below the feed shoots the solid shaft carries a collar with arms, to which doors are hinged. These doors may be either in a horizontal

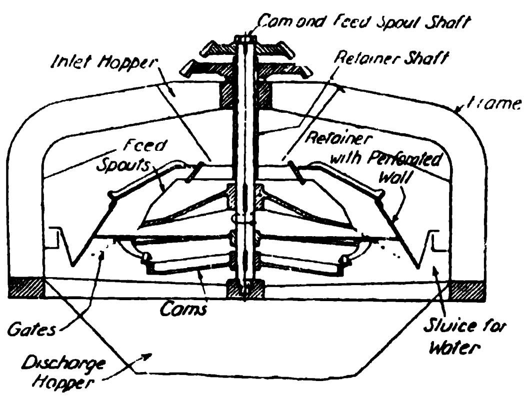

FIG. 234.—The Wendell Centrifuge.

position, when they almost touch the screen and hold the coal in the machine, or they may be opened by cams, fixed near the lower end of the solid shaft, to allow the coal to be discharged. The hollow shaft carrying the screen is driven slightly faster than the solid shaft carrying the feed spouts. As each door opens and discharges part of the contents of the machine, the following feed spout fills up the space again. The coal is therefore retained in the machine for a predetermined period and then discharged automatically.

In 1915 there were three machines in operation at the works of the Woodward Iron Company, where the washed coal was said to be dewatered from 30 or 40 to 8 per cent. water content (*Gas World*, 1915 (Coking Section, October), 18). The centrifuge is said to have passed through its experimental stages in 1910. One machine, in 1919, was treating 45 to 50 tons of coal per hour, using 18 to 20 h.p., although a 35 h.p. variable speed motor was used for starting (R. Gunderson, *Gas World*, 1919 (Coking Section, August), 11).

Raw coal, through a $\frac{1}{4}$ or $\frac{3}{16}$ in. screen, and containing 22 to 28 per cent. of water, was fed to the centrifuge and dewatered to 8 or 10 per cent. of water at a speed of 240 r.p.m. Monel metal screens were used.

DRYING BY HEAT

The term " dryers " is, strictly speaking, only appropriate when applied to those appliances by means of which coal can be completely dried, and not to appliances which only reduce the water content of coal or " dewater " it. All true dryers use direct or indirect heating and are generally of the rotating cylindrical type. They have been mostly employed for drying coal for pulverised-fuel firing, and are more suitable for this purpose than for dewatering washed small coal for coke manufacture. A rotary dryer of the Ruggles-Coles type has, however, been used in Germany for drying the concentrate from a coal froth-flotation plant. It will therefore be of interest to describe one or two types of these dryers and to compare them with approved types of centrifuge.

The Ruggles-Coles Dryer.—The Ruggles-Coles dryer has been used for drying a large number of minerals and heavy chemicals; its use for coal drying has so far been restricted to the particular conditions required for pulverised-fuel firing plants. The dryer was developed by the Ruggles-Coles Company of America, and was built in Great Britain by Electro-Metals, Ltd., and the Boving Engineering Works, Ltd., until 1924, when the European rights, except for the name " Ruggles-Coles," were acquired by Edgar Allen & Co., Ltd., Sheffield. The dryers are built to a number of different designs, but in all of them single or double cylindrical shells are used. Indirect heating at a low temperature by steam, or at a higher temperature by hot flue gases, are sometimes employed, and direct heating by hot air is the means adopted in another type. The highest drying efficiencies are obtained, however, by employing direct heating with hot flue gases, and it is this class of dryer which is used for coal. The coal dryer, illustrated in Fig. 235, consists of two long concentric steel plate cylinders set with the delivery end slightly lower than the upper feed end. The two cylinders are rigidly connected at their mid-lengths by six cast-iron braces (Fig. 236). Additional support is given by the provision of swinging braces at one-quarter lengths of the barrel, on either side of the fixed support, as well as at the lower end of the internal shell. This arrangement allows differential expansion of the two cylinders without the straining of joints. The outer cylinder has two steel tyres riveted to it, each of which is supported on four bearing wheels of chilled iron. The bearing wheels are arranged in pairs on rocker arms, which are supported on heavy cast-iron bases. One of the bases is also fitted with two thrust wheels, which operate on either side of one steel tyre to prevent longitudinal movement of the shells.

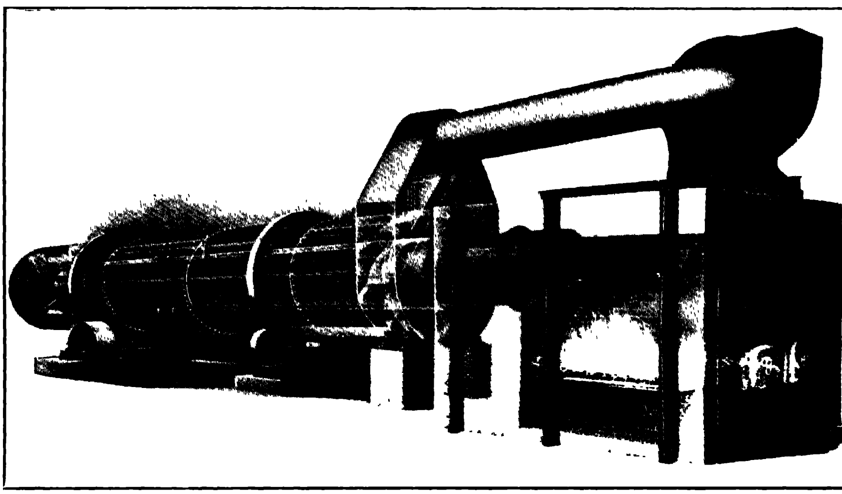

FIG. 235. The Ruggles-Coles Dryer—Phantom View

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A gear ring riveted to the outer shell near one of the tyres engages with a pinion on a cross (or a direct) drive.

Twelve lifting flights are equally spaced in the cross-section of the outer shell and are extended along the whole internal length. Six lifting flights are similarly fitted on the outside of the inner cylinder (see Fig. 236). The inner cylinder is extended at the upper end to receive hot flue gases from a furnace, and is brick-lined to prevent excessive wear and heat losses. The upper end of the annular space between the two cylinders is connected to a fan mounted on the top of the furnace. Through the fixed end casing a feed shoot supplies the wet coal to the annular space.

The hot flue gases, admixed with air, pass through the inner cylinder and return through the annular space to the exhaust fan. As the dryer rotates, the wet coal is lifted by the radial flights of the outer cylinder and is dropped on to the hot inner cylinder, where it is retained for about half a revolution. This operation is repeated a number of times as the coal travels to the lower end of the dryer. At the lower end, a series of buckets discharge it through a central delivery orifice. The wet coal is therefore subjected to indirect heating as the hot flue gases traverse the inner shell, but is directly heated by the cooler flue gases as they pass through the annular space to the fan. The temperature of the hot gases entering the inner shell is adjusted, by air addition, to a temperature of about 730° C., which is reduced to about 180° C. at the end of the inner cylinder, and to 65° C. at the exhaust fan.

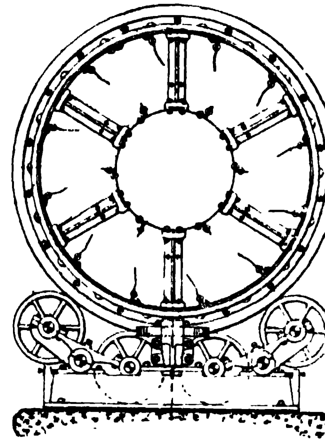

FIG. 236.—The Ruggles-Coles Dryer: Cross-section.

The largest dryer of this type has an external diameter of 7 ft. 6 in. and a barrel length of 55 ft., and a capacity of 20 tons of coal per hour. The over-all ground space occupied by the barrel and furnace is 74 ft. 2 in. by 12 ft. 4 in. The speed of revolution is 12 r.p.m. The evaporation efficiency is about 75 per cent. of the heat of the fuel. The power required to rotate the barrel and the elevators would be 20 to 25 h.p. for the size considered, with a further 5 h.p. to exhaust the very large quantity of steam and waste gas.

The Rhineland Dryer.—The Rhineland dryer (Fig. 237), consists of a single horizontal cylindrical shell which rotates about its axis; the interior of the drum is honeycombed by plates to form a series of communicating cells. Hot gases pass through the drum from end to end, and the feed coal is dropped from cell to cell, as it

is lifted by the rotation of the drum and falls by gravity. Intimate contact between the coal and the heating gases is therefore obtained, and practically all the cross-section of the drum is utilised for heat transfer.

The Büttner Dryer.—The Büttner cylindrical dryer is similar in some respects to the Rhineland dryer in dividing the cross-section of the cylinder into a number of cells. This is arranged by using five continuous stepped plates across the full width of the cylinder, with a cross-plate through the middle of each step. In an end view there are five rows, each made up of five + units. In addition, lifting arms are disposed on the internal periphery of the cylinder. The material fed to the dryer falls from one unit to another, and comes into direct contact with hot flue gases passing through the dryer. The waste gases are withdrawn by a fan and are forced through a cyclone dust extractor to settle the dust carried over.

The Büttner dryer was introduced in 1905, and since then over 600 units have been erected, mainly at sugar factories. It is also used for drying inorganic materials, and many units have been supplied for drying brown coal, lignite, bituminous coal and slurry.

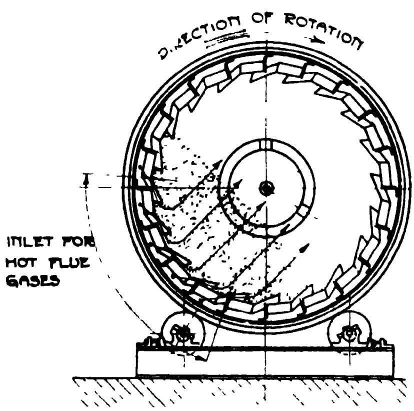

FIG. 238.—The Pehrson Dryer.

The Pehrson Dryer.—The Pehrson dryer has been used successfully in Sweden for drying fine charcoal. Its application to coal drying is now being examined in England on a semi-large scale. The dryer illustrated in Fig. 238 consists of a rotating horizontal drum, with an internal broken cylinder made up of a system of louvres, to which hot gases at a temperature of about 400° C. are supplied. The hot gases are forced through the louvres and through the coal and are exhausted by a second fan which discharges into a cyclone extractor to remove the dust carried forward.

The H. H. Dryer.—The H. H. dryer is cylindrical, but, unlike those previously described, is disposed vertically and is stationary. It has been called a gravity dryer, because the wet coal is fed in at the top and works its way down to the bottom partly through the agency of gravity. The cylinder is divided into a number of compartments, each of which consists of a horizontal tray with an inverted metal cone above it. Coal is fed into the cone of the upper

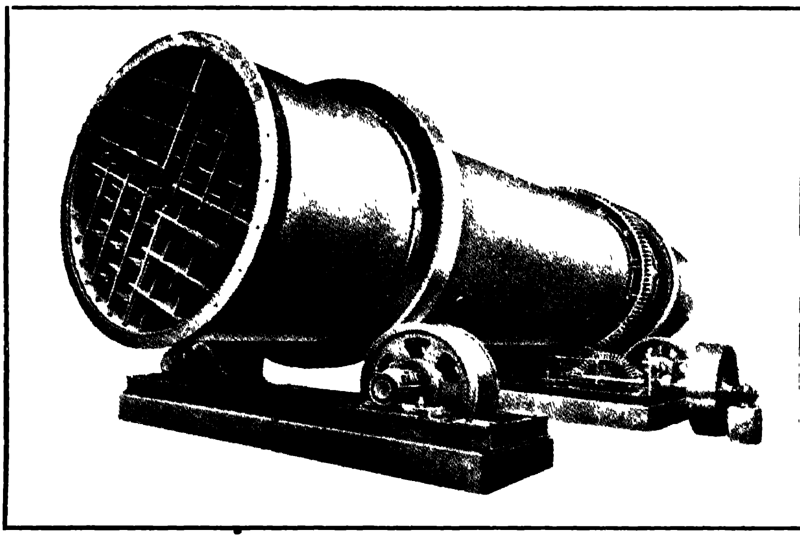

FIG. 237.—The Rhineland Dryer

[To face page 504

compartment and falls on to the first tray. Excessive breakage of coal, with its accompanying dust nuisance, is avoided by employing the cone to break the fall of the coal as it passes from one compartment to another. The coal is scraped by rotating rabbles from one tray to its outer edge, and falls into the cone of the compartment below, and so on. The cones are perforated, and a current of hot flue gases, passing up the dryer, comes into intimate contact with coal particles as they pass over each tray and drop from one compartment to another. The waste gases are discharged at a temperature above the dew-point through a cyclone. The dried coal is removed from the dryer by a worm conveyor.

The Lopulco Dryer.—The Lopulco vertical steam dryer was introduced to overcome some of the disadvantages sometimes experienced with dryers using hot flue gases, in which, it is said, risk of firing arose if the temperature of the flue gases rose unduly. One disadvantage of horizontal cylindrical dryers is the excessive ground space occupied. The Lopulco dryer consists of a stationary outer casing disposed with its axis vertically. An internal rotating frame carries sixteen circular tables, arranged in tiers, which nest one into another by suitable lugs, thus avoiding the use of bolted connections. Each table is made of cast iron, and has a set of steam coils cast inside it, the steam pipes being connected to inlet feeders and drainage openings in the central revolving frame. Steam is supplied to the inlet feeders through the hollow driving shaft, and, after passing through the table coils, escapes through the main drainage pipe. The inner revolving frame is driven by a pinion which engages on an internal rack fitted to a spider casting at the top of the dryer. Raw coal is fed to the dryer by means of a plough and a revolving table. The coal is spread uniformly over the first table by a spreader bar. Towards the end of one revolution the coal is scraped off into a shoot which discharges it on to the table below, where it is distributed uniformly by a second spreader bar. The coal passes from table to table, and is finally discharged from the bottom of the dryer. Air is admitted through perforations in one section of the outer casing of the dryer and passes over the coal on each table. The air is withdrawn from the dryer from above each table to a common collecting main and is discharged by a fan through a cyclone. By making separate air admissions to each compartment, the gas velocities are low, so that little dust is carried away. This dryer has been installed at a number of power stations where pulverised fuel is employed. The capacity is 7½ tons per hour, and 50 lb. of steam are used per 1 per cent. of water removed. The ground space occupied is about 13 ft. by 10 ft.

Comparing a direct heat dryer with a centrifuge for the same duty, the centrifuge would absorb, say, 40 h.p. in dewatering 40 tons

of coal from 12 to 7 per cent. of water. The ground space occupied would be about 10 ft. square. Ruggles-Coles dryers to handle the same amount of coal would require, say, 50 h.p. for driving the two barrels, and the fans. The fuel required, in addition to the power for driving, would be 512 lb. of coal (of 13,000 B.Th.u. per lb.), or 546 lb. of coke breeze (12,250 B.Th.u. per lb.) assuming 75 per cent. efficiency. The ground space necessary would be at least 75 ft. by 25 ft., and a separate building would be essential. It is clear, therefore, that for dewatering large quantities of coal for coking purposes, the Ruggles-Coles dryer cannot compete with an efficient type of centrifuge.

On the other hand, centrifuges have not been found to be serviceable for very small sizes of coal—slurry—which is the material for which dewatering is most essential. Slurry will retain, say, 30 per cent. of water even after dewatering on a jigging screen. The possibility of dewatering this material in a direct heat dryer is therefore worthy of consideration. Slurry containing 30 per cent. of water has not an appearance of wetness, but will, nevertheless, aggregate in lumps which cannot be evenly admixed with the rest of the coal.

To make slurry as useful as the larger sizes of a coking coal for coke manufacture it should be dewatered to the same extent, namely to a water content of not more than 10 per cent. The amount of slurry produced in a washery handling 100 tons of coal per hour varies from, say, 5 to 30 tons per hour, according to the friability of the coal. To treat 20 tons of slurry per hour, draining it to 30 per cent. of water on a jigging screen, and drying the drained coal by heating, to a moisture content of 10 per cent., the following fuel and power requirements would have to be met. Assuming a 75 per cent. efficiency, the 4 tons of steam produced per hour would require the burning of 1,024 lb. of coal slack, or 1,092 lb. of coke breeze. This is equivalent to the duty of a 30 ft. by 9 ft. Lancashire boiler, and gives an idea of the magnitude of the task. The volume of the steam (at 100° C.) and of the waste gases (at 50° C. and containing 8 per cent. of CO_2) would be 10,200 cu. ft. per min. The fan capacity necessary to handle this quantity would be very great, and 15 to 20 h.p. would probably be necessary for the fan itself. A big disadvantage of handling these large quantities of gases, with the consequent high velocities attained in the barrel, would be the possibility of carrying away fine material with the exhaust. The barrel and elevators would absorb 25 to 30 h.p., so that it is clear that at least 2 h.p. would be required per ton of wet slurry. A fuel consumption of approximately $\frac{1}{2}$ ton of coal (or coke breeze) per hour for the production of 16 tons of slurry containing 10 per cent. of water would also be required.

In spite of the large power and fuel requirements, the large ground space occupied and the capital and labour costs, it may well be that the adoption of such a scheme of dewatering slurry would

prove a profitable undertaking. The slurry problem would be definitely solved, the working of the ovens would not therefore be interfered with, the coking time would be reduced, and the output per oven increased, and less coke breeze would be produced. With turbine grates, coke breeze could be used as the fuel for drying.

DRYING BY OTHER METHODS

The Minerals Separation Process.—Another method of dewatering which has been devised for dewatering the product from a froth-flotation plant, is based on quite different principles to the ones already recorded. The method was described by L. A. Wood (*Proc.* Cleveland Inst. Eng., 1923, p. 13), and in B.P. 222,221.

The method makes use of the preferential wetting of coal surfaces by liquid hydrocarbons so that water films may be displaced from the coal surface and the water, being then only mechanically-entangled among the coal particles, may be displaced by pressure or suction. In the patent specification the hydrocarbons specified are oil, pitch and tar, which are agitated with the flotation product in amounts of 4 or 5 per cent., the product being then subjected to pressure. One form of apparatus used consists of two endless belts of coconut matting which are revolved by suitable pulleys. The upper surface of one belt and the lower surface of the other belt are in juxtaposition, and the coal, after oil treatment, is fed between the two belts. Pressure is applied to the coal by rollers fixed above and below the dewatering sections of the belt, by the adjustment of vertical rods on the sliding journals of the roller spindles.

At Aberaman, Wales, after agitation with a mixture of 5 per cent. of tar and pitch, coal was dewatered from 40 per cent. to 21 per cent. under a pressure of 7 lb. per in. of roller width. By increasing the pressure from 7 lb. per in. width of the first roller, to 14 lb. per in. at the fourth roller, coal was dewatered from 36 to 18 per cent. At Ashington, Northumberland, using only two rollers, coal was dewatered from 27 to 20 per cent. The product does not stick to the rollers and may be easily handled by an ovoid press.

By the application of higher pressures in presses the following results are said to be obtained.

Pressure (Tons per sq. in.).	Water per cent. in Product.
1	9·6
1½	8·2
2	6·9
3	5·8
4	5·3

CHAPTER XXVII

WASHERY WATER

General.—The water used for coal washing is usually collected by surface drainage into ponds or is pumped from pit workings. Town water is usually too expensive except in special circumstances. Some pit waters from deep colliery workings are heavily impregnated with mineral salts and are therefore unsuitable, but waters obtained by surface-drainage or from shallow colliery workings usually contain only small quantities of dissolved mineral salts, and are therefore satisfactory.

In a Baum washer of 75 tons per hour capacity, 120,000 gallons of water are required to fill the pipes and tanks of the washery system. In the washer of 150 tons per hour capacity, 230,000 gallons of water are required. When the washery system is filled with water, the pumps circulate water from the top of the settling tanks to the wash-boxes and the water returns from them by gravity to the settling tanks. In a Baum washer of 75 tons per hour capacity, the whole of the water in the system is circulated in an hour, or 2,000 gallons are circulated per minute (1,600 gallons per ton of coal). In the washer of 150 tons per hour capacity, about 3,300 gallons of water are circulated per minute, or 1,330 gallons per ton of coal. In a Rheolaveur washer of 60 to 80 tons per hour capacity, about 1,700 gallons of water are circulated per minute, or 1,260 gallons per ton of coal. In Rheolaveur washers of from 125 to 150 tons per hour capacity, about 2,250 gallons of water are circulated per minute, or about 920 gallons per ton of coal. In simple trough washers (of low capacity) the amount of water circulated may vary from 750 to 3,000 (or more) gallons per ton of coal washed. In Elliott trough washers from 1,600 to 2,000 gallons of water, and in Blackett washers 2,000 gallons (or more) of water are circulated per ton of coal. In washers of every type, make-up water must be added to compensate for the amount removed in the washed coal. This quantity is almost independent of the type of washer used and depends mainly on the fineness of the coal and the provision made for drainage. Generally, on leaving the primary dewatering screens or elevators, the total washed products (nuts, smalls and refuse) contain an average of about 10 per cent. of water, and from a washer treating 100 tons of coal per hour, about 10 tons of water will therefore be removed per hour. Make-up water at the rate of about 40 gallons per minute must be added, and may be admitted at the top of the

slurry-settling tank, or overhead water-supply tank, or as a spray on the washed-coal dewatering screens.

Water Supply.—The choice of water-supply for washing purposes is usually governed by financial considerations, but the water should not contain much suspended or dissolved material. Dissolved salts may corrode the metal work of the washery pumps or of the hoppers and wagons into which the coal is loaded. Moreover, contaminated water may damage the walls of coke ovens, and may spoil the appearance of washed coal when the water evaporates from the surface layers.

The analyses recorded in Table 134 refer to three possible sources of supply for a certain washery, namely, town's supply, water pumped from shallow colliery workings, and waste liquor collected in a pond. This last source was unsuitable and is only given for comparison.

TABLE 134.—ANALYSIS OF DIFFERENT WATERS

	Town Water.	Shallow Workings Water.	Waste Liquor.
Solids in suspension	Nil.	Nil.	Nil.
Dissolved solids (gm. per litre)	0·06	2·08	7·27
Degrees hardness—			
Temporary	1	34	3
Permanent	15	32	294
Total	16	66	297

Dissolved Salts.—The analysis of a water pumped from shallow colliery workings was given by Rees (Coke Oven Managers' Year Book, 1920). His figures are recorded in Table 135, together with an analysis of the water after it had been in circulation in a washery (in which the raw coal contained 0·37 per cent. of sodium chloride), as well as the analysis of the water draining from the washed slack hoppers.

Rees also suggested probable combinations of the constituents isolated: in Table 136 his results are calculated as a percentage of the total solids.

The principal dissolved salts in these waters were sodium chloride and sodium sulphate, which increased in concentration with continued use of the water. The fresh water contained 0·07 per cent. of chlorine, expressed as sodium chloride, the washery water in circulation 0·24 per cent., and the hopper drainings 0·30 per cent.

TABLE 135.—ANALYSIS OF WASHERY WATERS (GM. PER LITRE)

	Fresh Water.	Water in Circulation.	Drainings from Hoppers.
Lime (CaO)	0·154	0·095	0·102
Magnesia (MgO)	0·143	0·108	0·098
Sodium Oxide (Na_2O)	0·304	1·078	1·402
Potassium oxide (K_2O)	0·028	0·033	0·038
Chlorine (Cl)	0·274	0·928	1·180
Sulphuric anhydride (SO_3)	0·480	0·593	0·626
Carbonic anhydride (CO_3)	0·069	0·051	0·044
Iron oxide (Fe_2O_3), and alumina (Al_2O_3)	0·002	0·006	0·013
Silica (SiO_2)	0·012	0·014	0·018
Total solids	1·469	2·780	3·290

TABLE 136.—PROBABLE COMBINATION OF CONSTITUENTS IN WASHERY WATERS PER CENT.

	Fresh Water.	Water in Circulation.	Drainings from Hoppers.
Sodium chloride (NaCl)	32·3	57·9	59·0
Sodium sulphate (Na_2SO_4)	10·7	19·8	23·3
Magnesium sulphate ($MgSO_4$)	25·6	10·0	7·9
Calcium sulphate ($CaSO_4$)	17·3	6·4	5·0
Potassium sulphate (K_2SO_4)	3·7	2·3	2·2
Calcium carbonate ($CaCO_3$)	6·8	2·1	1·9
Magnesium Carbonate ($MgCO_3$)	3·6	1·5	0·7

The increase in the salt content is, of course, due to the extraction of soluble salts from the coal. Green (C.O.M.A. Year Book, 1920) records an example in which the dissolved solids in washery water increased from 0·16 to 0·70 per cent. in a week's washing, the increase being mainly due to the solution of sodium chloride. Extraction of salt by the washery water occurs chiefly with coals which are won from deep pits. In general, considerable salt accumulation occurs in washery waters in South Yorkshire, Derbyshire and Staffordshire, but in Durham (where the coals are usually won from shallower pits) the " salt problem " does not arise.

At some washeries, where fresh water supplies are not easily obtained, salt is allowed to accumulate in the washing water until it attains a concentration of 1·0 per cent. In others, it is not allowed

to accumulate to a concentration greater than 0·25 per cent., and when this arbitrary point is reached, the whole of the water in the washery system is run to waste (usually at the week-end).

Acid.—Other impurities which may be added to washery water during washing are ferrous sulphate and sulphuric acid, resulting from the oxidation in moist air of iron pyrites (marcasite). Iron pyrites occurs in coal in two forms, pyrite and marcasite. Pyrite, which has a specific gravity of about 5·1, crystallises in cubes and is only oxidised slowly in air. Marcasite, which has a specific gravity of about 4·8, forms rhombic crystals, and is oxidised comparatively rapidly in air. Marcasite is found in thin sheets in the cleat of some coals, and lumps of such coals may be partly covered with dull yellowish-brown layers of marcasite.

With such coals the washery water becomes acid, and considerable damage may be done to the metal work of the washery if the acid is not neutralised. In one case known to the writers, cast-iron pipes 1 in. thick were corroded until they became perforated or reduced to wafer thickness, necessitating complete renewal of the pipe system and the pump. The actual percentage of acid present was only of the order of 0·02 per cent., but this was sufficient in its cumulative effect to do the damage described.

The effect of weak sulphuric acid on cast iron was studied by immersing test pieces in acids of varying concentrations for a period of a week, the acid being renewed daily. The results of these tests are recorded in Table 137.

TABLE 137.—EFFECT OF SULPHURIC ACID IN WASHERY WATER ON CAST IRON.

Strength of Acid (per cent.).	Loss in Weight of Test-pieces (per cent.).
0 0000	0·017
0·0025	0·015
0·0050	0·005
0·0075	0·022
0·0100	0·062
0·0125	0·084
0·0150	0·091
0·0175	0·145
0·0200	0·257
0·0250	0·182
0·0500	0·452
0·1000	0·819

The loss in weight of the first two test pieces was due to rusting, the rust being removed in washing and drying the iron before weighing. The action of the acid is apparent in the subsequent tests. From these results it was obvious that corrosion occurred with the most dilute solutions of sulphuric acid, and, to overcome this,

sufficient lime was added to maintain the water alkaline. Occasional shovelfuls of lime were added as the coal was emptied from wagons into the raw coal hopper. Regular tests were made of the washery water and, if more than 0·002 per cent. of acid was found in successive tests, the amount of lime added was increased.

The acidification of washery water through the oxidation of marcasite is, fortunately, only a rare occurrence, but when a new type of coal is washed, the possibility of acid formation should be carefully watched.

Suspended Solids.—In continued use, the effective specific gravity of washery water increases owing to the suspension in it of solid particles which do not obey the "normal" laws of fall. Particles below a critical size fall in water so slowly that the slight agitation caused by the constant circulation of washery water containing them is sufficient to cause them to remain in suspension. It has previously been pointed out that in ordinary practice this leads to less efficient washing. If the washed small coal, on passing to a drainage hopper, contains, say, 20 per cent. of dirty water, further contamination of the washed coal results, for the dirt is filtered out by the coal whilst the water drains away.

When washed coal containing 20 per cent. of water passes to drainage hoppers, the solids filtered out of the washing water may increase the weight of the coal by 2 per cent. If the solids consist of coal and dirt in equal proportions, the ash content of the coal would probably be increased by 0·5 per cent. solely on account of the contamination caused by the dirty water. Moreover, the finely-divided dirt material, which spreads over the coal particles, hinders drainage. For this reason, it is a common experience to find that the water content of the washed fine coal is greater at the end of a week, when the water is dirty, than at the beginning, when it is clean. This defect might be remedied entirely if sufficient fresh water were available to spray the washed coal as it passes along a drainage conveyor or over the drainage screen. In coking practice, where the added water increases the coking time in the ovens, this use of fresh spray water would usually be profitable, even when the cost of frequent water replacement cannot be faced.

The figures in Table 138 show the increase in the amount of solids in suspension in the washery water during an eight-hour washing shift with a Baum washer.

It will be seen from these figures that the percentage of total solids in the washery water leaving the top of the settling tank increases from hour to hour with the duration of working, and that the nature of the solid particles in the water changes, so that it contains a greater proportion of clay or dirt material. The figures recorded in Table 139 refer to another Baum washer. The percentages of solid material in the water entering and leaving the settling tank were noted during a six-day washing test.

TABLE 138.—CONTAMINATION OF WASHERY WATER DURING AN EIGHT-HOUR SHIFT

Time from starting (hours)	0	4·5	7·5
Per cent. of total solids in water	1·24	3·50	5·4
Dirt, per cent. of total solids	11·3	48·6	65·3

TABLE 139.—CONTAMINATION OF WASHERY WATER DURING A SIX-DAY PERIOD

Day.	Per cent. of Solids in Water.		Per cent. of Solids Settling.
	Entering Settling Tank.	Leaving Settling Tank.	
1st	2·80	2·50	10·7
2nd	6·56	6·50	1·0
3rd	8·37	7·50	10·4
4th	8·48	7·75	10·9
5th	13·00	8·45	35·0
6th	15·00	9·45	37·0

It will be seen from these figures that the percentage of solids in washery water increases with continued working; after the second day, however, the increase in the percentage of solids in the water actually used in the washer (leaving the settling tank) is only slow. A point of particular interest is the fact that very little of the solids settle from the water until the concentration is over 10 per cent., when considerable quantities are removed.

Drakeley (*Trans.* Inst. Min. Eng., 1917-18, 54, 457) records figures for an examination of fifteen samples of washery water taken from different washeries in the Lancashire coalfield. The specific gravity of the washery water varied from 1·009 to 1·138, and the water contained from 1·7 to 39·3 per cent. of solid particles (slime) in suspension. The specific gravity of the filtered washing waters varied from 1·001 to 1·007, showing that dissolved salts were present in all of them. The percentage of ash in the recovered slimes varied from 16·0 to 45·7 per cent. The average results for the fifteen samples were as follows:—

Specific gravity of the washing water	1·036
Specific gravity of the filtered washing water	1·002
Per cent. of slime in washing water	10·31
Specific gravity of the slime	1·58
Percentage of ash in slime	26·9

These slimes do not readily settle in ordinary slurry-settling tanks whilst the washer is working. When the washer is standing, some of the slimes settle, and may be removed from the washing system, but sometimes several days are required to effect a complete clarification of the water. For this purpose the washing water may be removed from the washery and run into settling ponds. Other methods of clarification are the use of rotary vacuum filters (Chapter 26) and the addition of flocculating materials.

CLARIFICATION

Washery water is usually clarified in spitzkasten (Humboldt, Rheolaveur, etc.), or in an inverted conical tank (Baum). In all these appliances the speed of movement of the water is reduced to allow suspended particles to settle.

In the elevated conical settling tank of a Baum washer (p. 161) the slurry settles to the bottom of the cone and is run off continuously. The concentration of the thickened slurry removed from a Baum settling tank at different periods of an eight-hour washing day is given in Table 140.

TABLE 140.—SLURRY REMOVED FROM BAUM SETTLING TANK

Time from starting (hours) . .	0	4·5	7·5
Solids in concentrated slurry (per cent.)	44·3	22·1	41·3
Dirt, per cent. of total solids . .	16·7	30·6	40·0

It will be observed that the proportion of dirt in the settled slurry increases the longer the washer is working.

The Dorr Continuous Thickener.—The Dorr thickener was originally devised to clarify solutions obtained in the cyanide process for gold recovery and has been adapted to the clarifying of washery waters. An example of the thickener illustrated in Fig. 239 consists of a cylindrical tank in which ploughs or scrapers driven by a central shaft move the settled solids to a central discharge opening. The feed passes from the trough (1) to a central feed well, which is surrounded by a circular plate to minimise the disturbance of the contents of the tank. The heavier solid particles settle to the bottom of the tank, and the clearest water from the top overflows into a launder round the periphery of the tank and is collected by the discharge pipe (3). A central shaft, actuated through suitable gearing, carries four radial arms fitted with inclined scrapers which

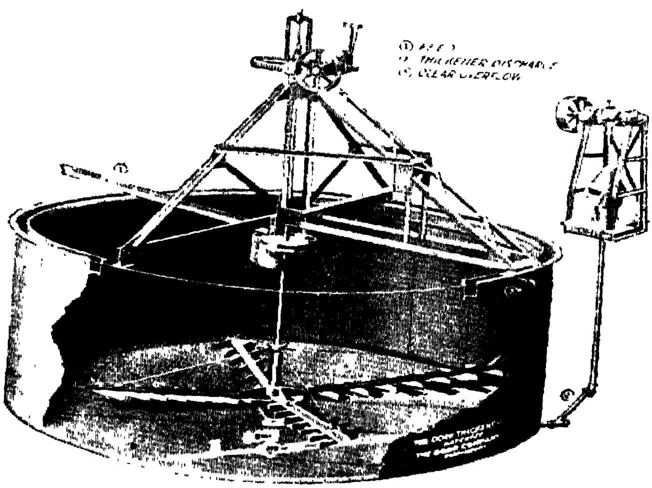

FIG. 239.—The Dorr Thickener.

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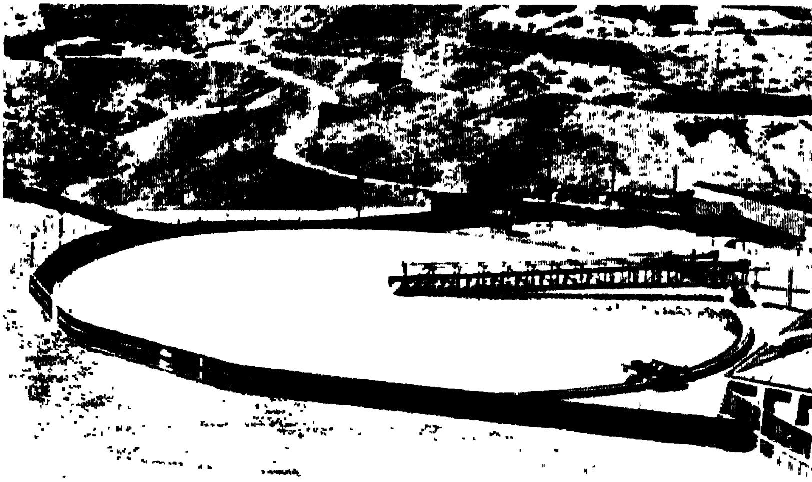

FIG. 240.—The Dorr Traction Thickener.

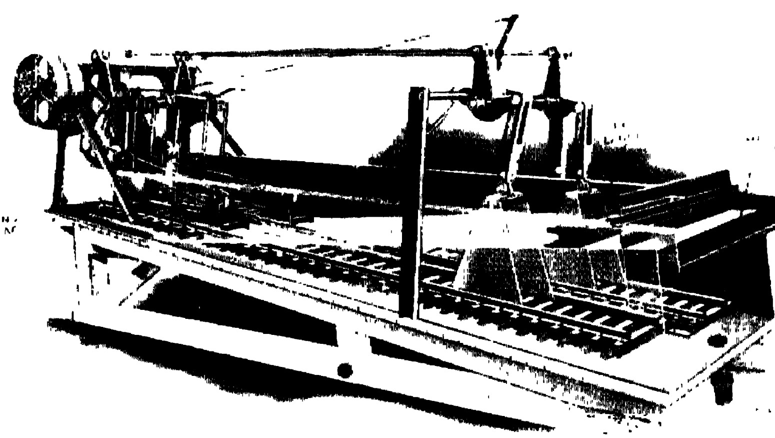

FIG. 241.—The Dorr Classifier

move the settled solids towards the centre of the tank. From the central opening in the bottom of the tank the thickened solids are withdrawn through the pipe (2) by means of a special suction pump (Dorrco rubber-diaphragm pump).

The tank illustrated is made of steel plates, but for coal-washing purposes the tank is usually built of wood or concrete. A simple excavation in the ground with concrete overflow rims is sometimes used. If the thickened solids accumulate unduly an overload alarm attracts attention. The scraping arms may be raised from the bottom of the tank if desired.

In the Dorr traction thickener a single scraping arm is used, and is pivoted at the centre of the tank and supported at its circumference by a carriage which moves round the tank on a track. The carriage is driven by a motor through suitable reduction gear. The scraper blades of the single arm are said to give the same scraping service as the four-arm type for the same diameter of tank because of their special length and depth. The slurry water is fed to the centre of the tank, unless the motor is overloaded or fails, when an electrically-operated by-pass device comes into operation. If the speed of the driving carriage is reduced by a partial overloading, an alarm bell sounds. A view of a large traction thickener is given in Fig. 240. This type of clarification pond has been built up to 200 ft. in diameter.

The radial arms make from 1 to 30 revolutions per hour, according to the fineness of the material, the speed being slower, the finer the particles. The power required for tanks up to 100 ft. diameter is said to be from 2 to 5 h.p. in starting, but less than this in operation.

A tank 50 ft. diameter and 8 ft. deep will deal, ea h hour, with 6,000 gallons of water containing 3,000 to 6,000 lb. of solids (5 to 10 per cent.). With a tank of 200 ft. diameter, 5,000,000 to 6,000,000 gallons of water containing 3,000 to 4,000 tons of solids can be treated per day. The power requirements for this capacity are 3 h.p.

The Dorr Classifier, illustrated in Fig. 241, is also used for the separation of coarse solid particles and fine solid particles suspended in water. The tank is simply a wide trough, suitably inclined and provided with steep sides. The upper end is open, and is above the water level in the tank. Slurry water is admitted near the closed end, and water with the finer solid particles overflows at the closed end. The coarser particles settle, and are moved up the inclined bottom of the trough by rakes and are discharged from the upper end. The rakes are actuated by a cam and bell-crank mechanism which drags the rakes through the settled solids on the bottom of the tank during one stroke, and on the return stroke lifts them above the layer of settled material.

The use of the Dorr classifier in ore-dressing practice is illustrated by figures recorded by Wiard (" Handbook of Chemical Engineering," Vol. I, p. 274, New York, 1922). A classifier 14 ft. 8 in. long and 36 in. wide dealt with a feed containing 610 tons of material mixed with about half its weight of water. The slope of the classifier was 3 in. per foot and the rakes made thirty-four strokes per minute. From the material fed, 330 tons of sand (dry) were recovered per twenty-four hours, mixed with water amounting to one-quarter the weight of the dry sand, and 280 tons of slimes mixed with about an equal weight of water. The feed solids would all pass through a 10 mesh screen, and 35·2 per cent. was less than 200 mesh size. The recovered sands contained only 5·1 per cent. of material which would pass through a 200 mesh screen, but all the slimes would pass through a 35 mesh screen and 77·6 per cent. through a 200 mesh screen.

The Dorr thickener may be of use in separating coal slimes from slurry The capacity of one unit would not be so great for coal slurries as for ore sands, since the rate of fall of small coal particles is less than that of sand particles of the same size. The finest sizes of slurry are usually the dirtiest, as shown by the following figures (Table 141).

TABLE 141.—FLOAT AND SINK ANALYSIS OF SOUTH YORKSHIRE SLURRY

Size (Mesh I.M.M.)	Per cent by Weight.	< S.G. 1·48.	> S.G. 1·48.
> 30 . .	5·3	5·0	0·3
30–60 . .	25·8	23·9	1·9
60–90 . .	14·5	12·0	2·5
90–120 . .	3·7	3·0	0·7
120–150 . .	14·1	9·8	4·3
150–200 . .	8·1	4·5	3·6
< 200 . .	28·5	4·1	24·4
Total . .	100·0	62·3	37·7

The floats of the fraction less than 200 mesh had an ash content of 2·2 per cent.; the sinks, 43·0 per cent.; and the combined floats and sinks, 37·0 per cent. If the through 200 mesh fraction (28·5 per cent. of the total) could be removed by sieving, only 4·1 per cent. of useful material would be lost, and the ash content of the slurry

would be reduced from 18·6 to 8·5 per cent. Direct sieving for this fine size would not be commercially possible, but the separation might be made in a Dorr thickener designed to float

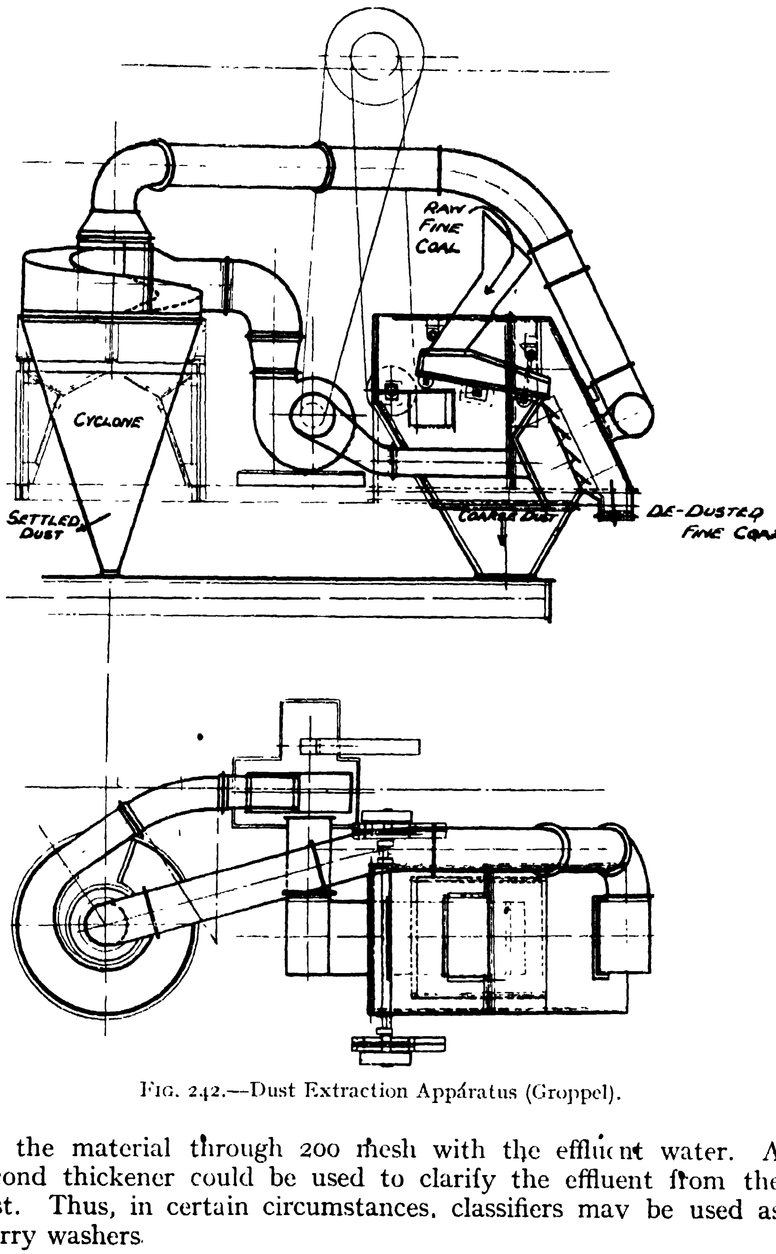

FIG. 242.—Dust Extraction Apparatus (Groppel).

off the material through 200 mesh with the effluent water. A second thickener could be used to clarify the effluent from the first. Thus, in certain circumstances, classifiers may be used as slurry washers.

CHAPTER XXVIII

FEEDING DEVICES, ELEVATORS, CONVEYORS, AND PUMPS

FEEDING DEVICES

THE usual method of ensuring a constant rate of feed of coal to a washery is to deliver the raw coal from an elevator into a shoot immediately preceding the washer. The elevator buckets may scoop the coal from a hopper at a sufficiently constant rate, but it is usually desirable to have an automatic means of feeding the elevator. When, however, it is not convenient to feed the coal from an elevator, some automatic feeding device is required to ensure a constant rate of supply to the washer.

The commonest form of feed control consists of a gate or of sliding doors operated by a rack and pinion. To prevent choking the aperture should be rectangular, circular apertures seldom being satisfactory.

Alternatively, the coal may be fed by a rotating table. In the simplest type of rotating table feeder, the coal is delivered from the hopper to a rotating plate (Fig. 243, A), frequently inclined slightly to the horizontal. The supply of coal to the revolving plate is adjusted by a gate. In another form, the coal is contained in a conical hopper closed at its lower end (the apex) by a serrated cone-shaped block attached to a circular rotating table. The taper of the hopper ceases about 1 ft. from the bottom, and the lowest portion consists of an adjustable annular ring. The ring can be set in different positions relative to the serrated cone-shaped seal, giving different widths of the annular spaces between the two. As the cone-shaped seal rotates, the coal falls on to the revolving table at a uniform rate, and is scooped off by a stationary arm set at an angle to the radius of the table. The rate of feed may be varied by adjusting the position of the annular ring.

When the coal is fed directly to the washer, some form of shaking tray or apron feeder is the most satisfactory, though plunger feeders, screw feeders and star feeders may be used.

An apron feeder consists of a travelling belt driven round drums by means of gearing. The belt usually consists of chain belting, or of castings riveted to a chain frame. The belt collects the coal from the base of the hopper (Fig. 243, B) and delivers it to a belt or other conveyor, or to a shoot. This type of feeder is most satisfactory with unsized material.

The Ross feeder (Fig. 243, C) is a form of apron feeder, the apron

consisting of heavy chain belt which supports a series of heavy bars The chain belt is driven by a sprocket or drum at its upper end When the drive is stopped, the bars act as a seal for the hopper.

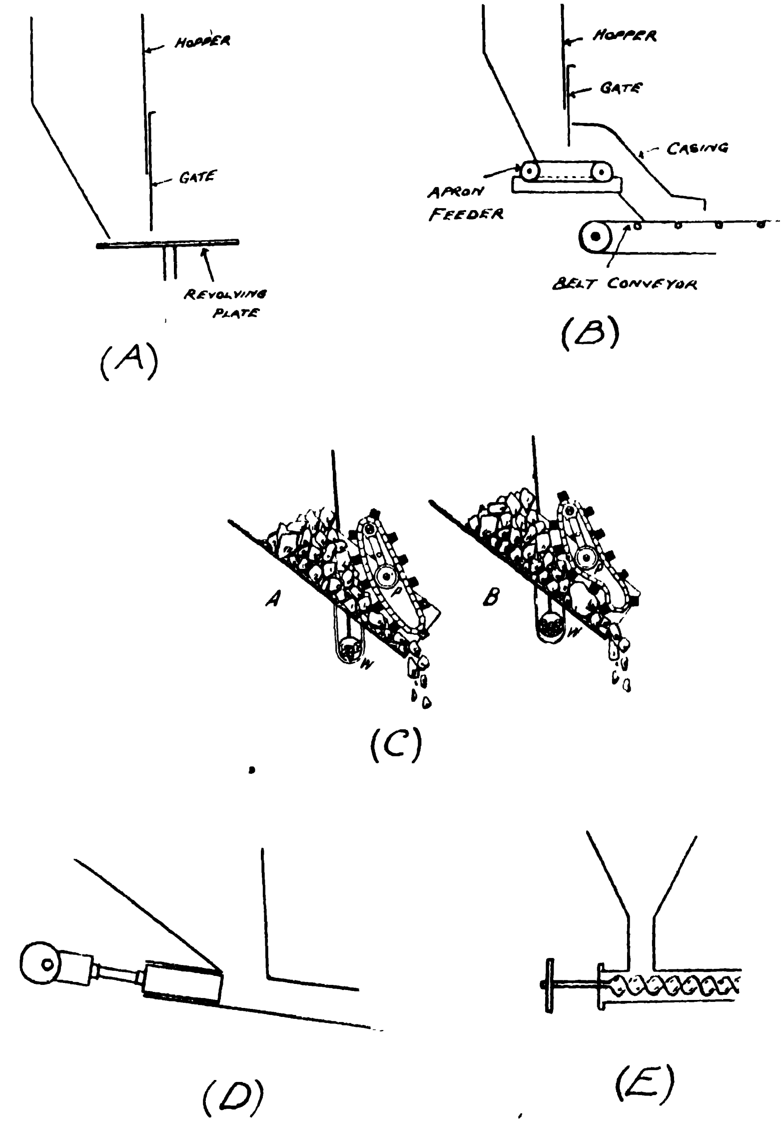

FIG. 243.—Feeding Devices.

When the bars are driven, however, the coal is discharged from the hopper at a rate which varies with the position of the balance weight, W. With an extra large piece, the bars are thrust outwards (as in B) and the weight is lifted from its seating.

With a plunger type of feed (Fig. 243, D) a piston driven by an eccentric works under the hopper and forces the material into a shoot. Feeds of this type are unsuitable for friable materials.

A screw feed is shown diagrammatically in Fig. 243, E.

ELEVATORS

Elevators are an important feature of nearly all coal washeries, and are used particularly to raise the unwashed coal to a suitable height for admission to the wash-boxes (raw-coal elevators) and to raise the washed coal from a drainage sump or dewatering screens to the drainage or storage bunkers (washed-coal elevators). They may also be used to raise the fine coal, middlings or "rewash" material to a sufficient height for feeding to a rewash plant, and to

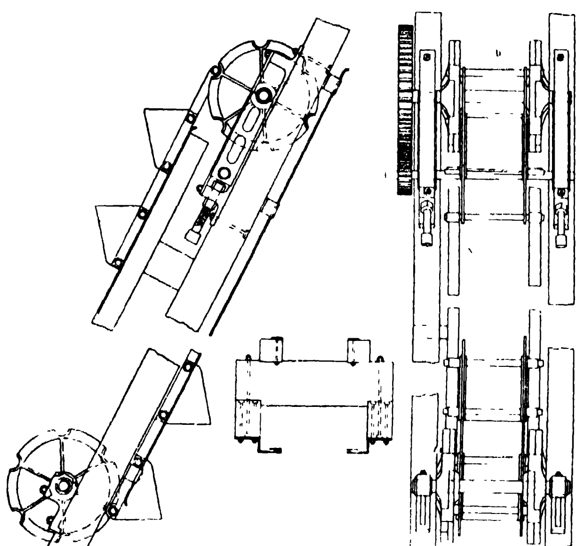

FIG. 244.—Details of Elevator.

feed the refuse to a discharge hopper. The capacities may vary from 10 to 150 tons per hour.

Elevators consist of a series of "buckets" of cast or wrought iron set between and secured at intervals to endless chains formed of flat links. For convenience of construction and for strength the links are placed on edge. For small elevators single-link chains are used, but for larger ones the chains are built up of single and double links alternately. The links are united by rods, bolts, cotter pins or rivets, which are turned to fit accurately into the holes bored near the extremities of the links. Two sets of link chains are joined by the link-connecting rods, and to the connecting rods the buckets are fastened. The chains are stretched over a pair of

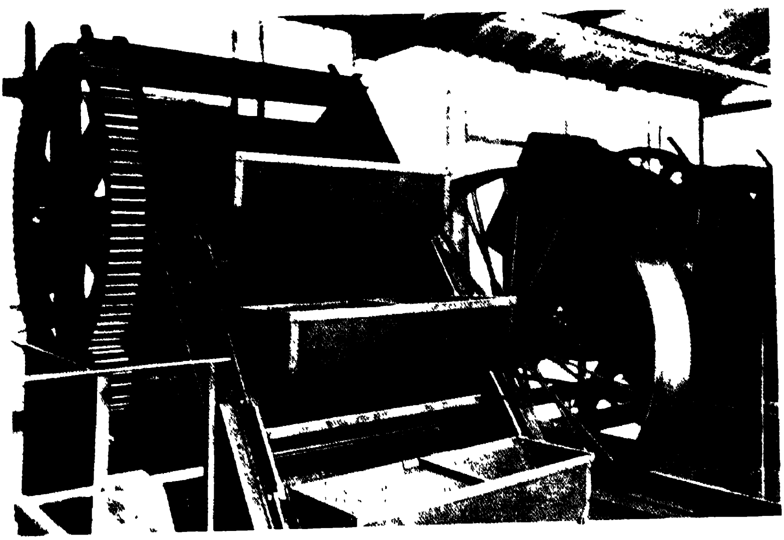

FIG. 245.—View of Top of Elevator

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polygonal drums mounted on revolving shafts. The shafts are supported by bearings secured to a strong framework. The elevator is usually inclined at 60 to 70 degrees to the horizontal. The uppermost drum is driven comparatively slowly by toothed gearing.

In certain cases the lower drum is used as the driver, but this necessitates greater tension on the chain to prevent the drums from slipping. In the usual method of using the upper drum as the driver, the tension due to the weight of the chain and the loaded buckets keeps the chain in contact with the driving drum. Provision is made for adjusting the distance between the centres of the drums to take up extension. In adjusting the tension on an elevator-chain, one set of bearings has to be moved ; some firms prefer to make the upper bearings the movable ones, keeping the lower ones fixed so that the buckets always pass close to the bottom of the elevator pit and scoop out the coal ; other firms prefer to make the lower bearings movable on account of the very heavy load on the upper bearings. When the bottom bearings are movable, provision should also be made to move the bottom of the elevator casing.

The links are supported on angle-iron slides and fixed frames, the links projecting a suitable distance below the buckets to allow for wear and to prevent the buckets from coming into contact with the slides. In Fig. 244 various details of an elevator and its casing are given, and in Fig. 245 a view of the top of an elevator with alternate double and single links.

Bucket elevators are loaded by making the buckets pass through the elevator boot, an underground hopper, or a sump, containing the coal or refuse. Gate-valves governed by a spindle and hand wheel, or mechanically-operated sliding doors, or rotary feed tables are sometimes used to regulate the entry of material to the buckets and to prevent them from becoming overloaded. The inclination of the elevator permits the free discharge of the contents of the buckets at the top of the elevator into a shoot.

The speed of an elevator is given by the formula :—

$$V = lnr,$$

where V = speed in feet per minute,
l = length of one link (between bolt centres),
n = number of sides on the polygonal drum,
r = number of revolutions per minute of driving drum per minute.

The capacity of an elevator is given by the formula :—

$$Q = \frac{wb}{37 \cdot 3}, \text{ or } Q = \frac{wV}{37 \cdot 3s},$$

where Q = capacity in tons per hour,
w = weight of the contents of one bucket in pounds,
b = number of buckets filled per minute,
s = pitch or spacing of buckets in feet.

Buckets may be fitted to every link of the elevator chain or to alternate links. When fitted to alternate links they are usually roughly V-shaped (Fig. 246, I), but when fitted to every link the width of the top of the buckets is often increased, and the outer sides of buckets make a bigger angle with the elevator casing (Fig. 246, II).

In Table 142 the capacities of elevators with different sizes of buckets are given (Gröppel, Kohlenaufbereitung). The buckets are assumed to be full and to be moving at a speed of 60 ft. per minute.

The power consumption of elevators is directly proportional to

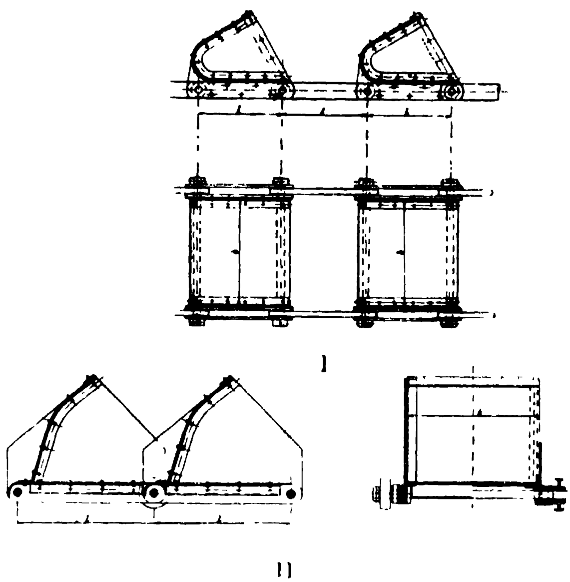

FIG. 246.—Details of Two Forms of Elevator Bucket.

the weight of the load carried and to the height to which it is raised. Allowance has also to be made for the frictional resistance, and for the power necessary to drag the buckets through the charge in the elevator boot. These factors increase with the size of the elevator. The general equation for power consumption is :—

$$\text{H.P.} = x\text{H},$$

where x = a factor to allow for varying frictional resistances,

H = height, in feet, to which load is raised.

Where the elevator is less than 50 ft. long, an addition (z) has to be made. In Table 143 (adapted from Gröppel, *loc. cit.*) values for x and z are given for varying capacities. (The numbers of the elevators are those given in Table 142.)

For intermediate values of Q, intermediate values of x are chosen.

TABLE 142.—CAPACITY OF ELEVATORS WITH DIFFERENT SIZES OF BUCKET

No.	Effective Length of Link and Bucket.*		Capacity of each Bucket (lb.).	Elevator Capacity tons/hour.	
	mm.	in.		Type I.	Type II.
1	0·2	$7\frac{7}{8}$	8·9	10	—
2	0·25	$9\frac{7}{8}$	15·4	15	—
3	0·30	$11\frac{7}{8}$	27·6	22	$39\frac{1}{2}$
4	0·35	$13\frac{3}{4}$	37·5	$25\frac{1}{2}$	$57\frac{1}{2}$
5	0·40	$15\frac{3}{4}$	62·9	$37\frac{1}{2}$	$74\frac{1}{2}$
6	0·45	$17\frac{3}{4}$	81·6	$43\frac{1}{2}$	87
7	0·50	$19\frac{3}{4}$	117·9	56	$112\frac{1}{2}$
8	0·55	$21\frac{5}{8}$	165·2	72	144

TABLE 143.—FACTORS TO CALCULATE THE POWER CONSUMPTION OF ELEVATORS (H P. — xH)

No. of Elevator.	Capacity (Q) tons/hour.	Power Factors	
		x	z (H.P.).
1, 2	5	0·077	0·5
1, 2	10	0·092	0·6
2, 3	20	0·124	0·7
3, 4, 5	30	0·154	0·8
4, 5, 6	50	0·186	0·9
6, 7	75	0·201	1·0
7	100	0·248	1·1
7	125	0·295	1·2
8	150	0·342	1·3
8	200	0·418	1·4

* Length of bucket is the length on the link chain. For simplicity breadth of bucket (b, Fig. 246) is taken as being equal to the length (l). In practice, the breadth is usually greater, and due allowance should be made for this, in applying the figures given. When elevating wet materials the capacity, Q, should be multiplied by 0·8. With special shapes of buckets the capacities given can be increased by 30 per cent.

The power necessary to work the empty elevators is given by the equation

$$H.P. = \frac{QH}{900},$$

where Q = the nominal capacity of the elevator in tons per hour,
H = the height in feet of the elevator.

For example, a raw coal elevator raising 150 tons of coal per hour to a height of 80 ft. would require

$$80 \times 0{\cdot}342 = 27{\cdot}4 \text{ h p.}$$

A dirt elevator raising 20 tons of material to a height of 20 ft. would require

$$20 \times 0{\cdot}124 + 0{\cdot}7 = 3{\cdot}18 \text{ h.p.}$$

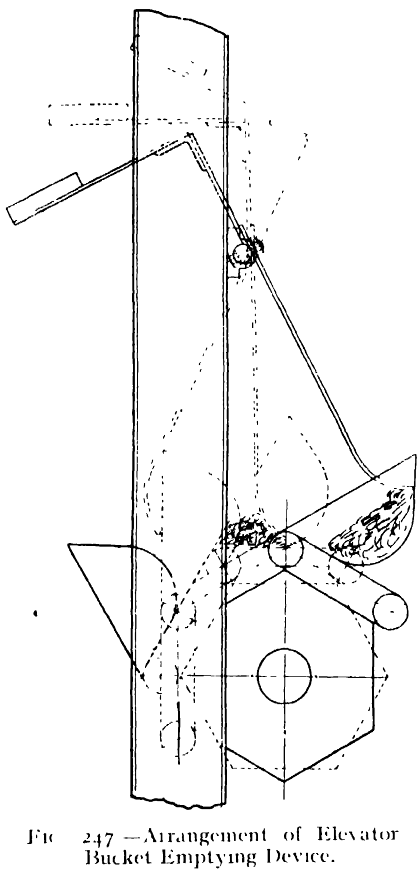

FIG. 247.—Arrangement of Elevator Bucket Emptying Device.

To these calculated horse-power requirements an addition should be made to allow for starting, and to provide a margin for over-loading.

Much of the maintenance cost of a washery is expended in repairs and renewals to the elevators. The chief source of expense is in the renewal of links and of the rubbing strips. Adjustment of the bearings to keep the necessary tension of the chains to prevent them from slipping on the drum heads is also a frequent source of trouble. E. T. Hardy (Coll. Eng., ...) recommends wagon-spring steel ⅜ in. thick for rubbing strips. He also advises the use of planished steel bar for link pins, with split pins inserted through the collars and through the link pin to secure it. Bolts, ⅝ in. diameter, at 15 in. intervals, are recommended to secure the rubbing strips to the elevator casing, the bolt heads being at least ⅛ in. below the surface of new rubbing strips. Wooden rubbing strips are also serviceable

With fine wet materials, difficulty is often experienced in the free discharge of the buckets of an elevator, especially when the material is refuse from fine coal or slurry washing. This difficulty was experienced with the refuse elevator in the Yorkshire Coking and

Chemical Company's Rheolaveur slurry washer at Glasshoughton, and a scooping device was designed to overcome it.

As the bucket is about to tip over, a steel plate is caught by the outer face of the bucket, and as the bucket moves further round its circumferential path the plate travels into the bucket down its outer face and under the bed of material in the bucket. When the bucket is inverted its edge bears against the plate, and forces it out of the bucket, the contents being removed by the plate. The plate is attached to a hinged and weighted lever so that it is automatic in action. The device is illustrated in Fig. 247.

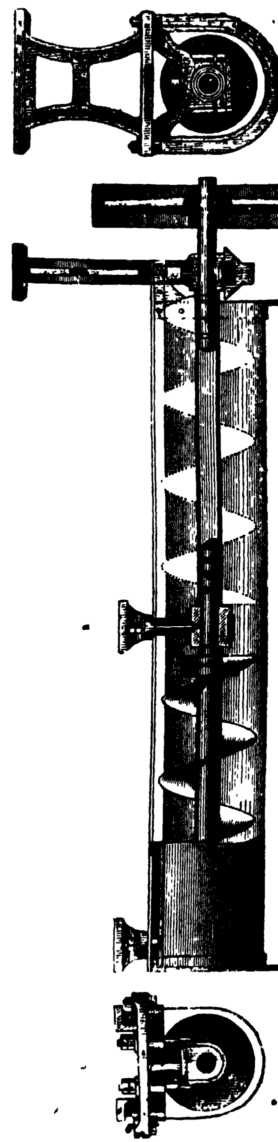

FIG. 248.—Screw Conveyor.

CONVEYORS

Screw Conveyors.—For transport of coal or dirt for short distances on the level, screw conveyors are often used. A screw conveyor consists of a shaft supported by journals at intervals and having a screw of coarse pitch and thin thread working in a trough or pipe of slightly greater width than the diameter of the screw (Fig. 248). Intermediate journals revolve in steps fixed in hangers suspended from the top of the trough, and the end journals revolve in steps in ordinary plummer blocks, provision being made for end thrust. The screw is driven by a pulley or toothed wheel fixed on the end of the shaft outside the trough, usually at the feed end of the conveyor. The load is discharged from a screw conveyor through bottom openings in the trough, or through the open end of the trough. Screw conveyors may also work in water. The capacity of a screw conveyor depends on the diameter, pitch, and speed of rotation of the screw. The capacity is given by the equation :—

$$Q = \frac{apn}{128},$$

where Q = capacity in tons per hour.
a = area of cross-section of trough filled by material (in square inches).
p = pitch of screw in inches.
n = number of revolutions per minute.

The trough is usually filled for about $\frac{1}{3}$ of the cross-section of the trough. The figures recorded in Table 144 for the capacities and power consumptions of screw conveyors are adapted from Gröppel (*loc. cit.*). The power consumption is given by the equation :—

$$H.P. = Lx,$$

where L = length of conveyor in feet,
x = a factor obtained from table,
z (see Table 144) is an addition to be made to the horse-power when the conveyor is less than 10 metres (32 ft. 10 in.) long.

TABLE 144.—CAPACITIES AND POWER CONSUMPTION OF SCREW CONVEYORS

Diameter (in.).	Pitch (in.).	Speed r.p.m. (n).	Capacity (Q) Tons/hour.	Horse Power Factors.	
				x	z (h.p.).
$7\frac{7}{8}$	$7\frac{1}{8}$	65	5·8	0·076	0·3
$9\frac{7}{8}$	$7\frac{7}{8}$	55	8·6	0·092	0·4
$11\frac{7}{8}$	$9\frac{1}{2}$	48	12·8	0·14	0·5
$15\frac{3}{4}$	$12\frac{1}{2}$	36	22·8	0·18	0·6
$19\frac{3}{4}$	$15\frac{3}{4}$	30	37·1	0·27	0·7
$23\frac{5}{8}$	$17\frac{3}{4}$	25	50·2	0·37	0·8
$27\frac{1}{2}$	$19\frac{3}{4}$	21	57·2	0·41	0·8
$31\frac{1}{2}$	$23\frac{5}{8}$	18	76·8	0·49	0·8

The pitch is approximately 0·8 times the external diameter.

Scraper Conveyors.—A creeper or scraper conveyor is used for the transport of coal on the level to feed, say, a number of storage hoppers. It may also be used to raise coal to considerable heights. It is, perhaps, the most convenient form of conveyor for supplying any one of a series of hoppers arranged in a row, for the floor of the scraper can include a trap-door, which may be opened to form an aperture or closed to form part of the floor. A scraper or drag conveyor consists of a double endless chain, formed by links, stretched

over a pair of polygonal drums at each end of the frame, and is supported between the drums by rollers or slides. The links are

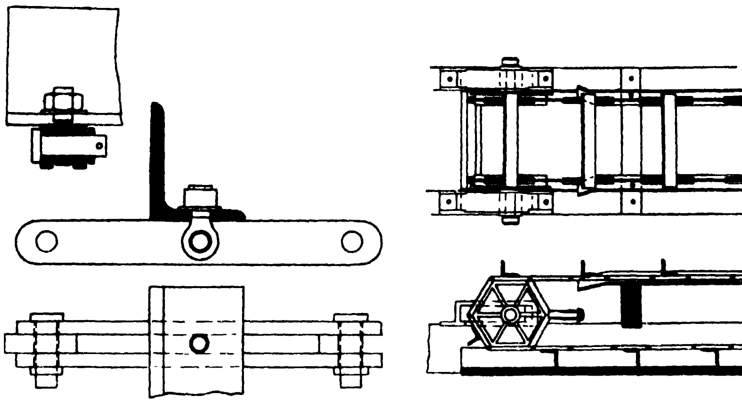

FIG. 249.—Details of Scraper Conveyor.

joined together by pins which are extended to form scraper plates between the two chains, moving over a solid plate surface. Renewable lining strips are fitted to the base plates. Various details of a scraper conveyor are given in Fig. 249.

Belt Conveyors.—Belt conveyors are frequently used to transport coal to a washery. A belt conveyor usually consists of an endless rubber-faced canvas belt running about end pulleys and supported on the run by a series of idle rollers which are often " troughed " on the conveying run and are horizontal on the return run. Sometimes horizontal rollers are used on the conveying run. With both kinds of support, the individual rollers are of uniform diameter to ensure contact with the travelling belt across its whole width. On the return run, the rollers are mounted on a common shaft, or a single roller slightly wider than the width of the belt is used. The return rollers are spaced at about 10 ft. intervals, and where the load is carried, at intervals of from 2 to 5 ft. The spacing depends on the width of the belt, the intervals being reduced with wider belts. At loading points the spacing is reduced to about two-thirds of the intervals used on conveying stretches, and the belt is supported about 6 in. behind the loading shoot to give added strength. For conveyors

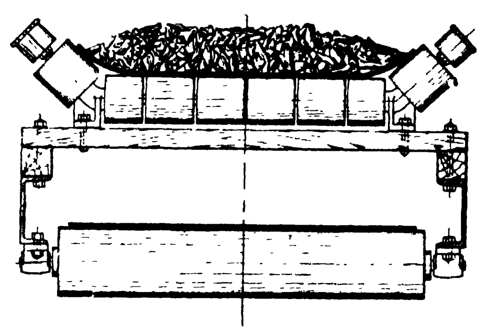

FIG. 250.—Belt Conveyor Supporting Rollers.

longer than 25 ft., guide rollers mounted normally to the edges of the upper belt are recommended. An arrangement of idle rollers for the conveying and return runs is illustrated in Fig. 250,

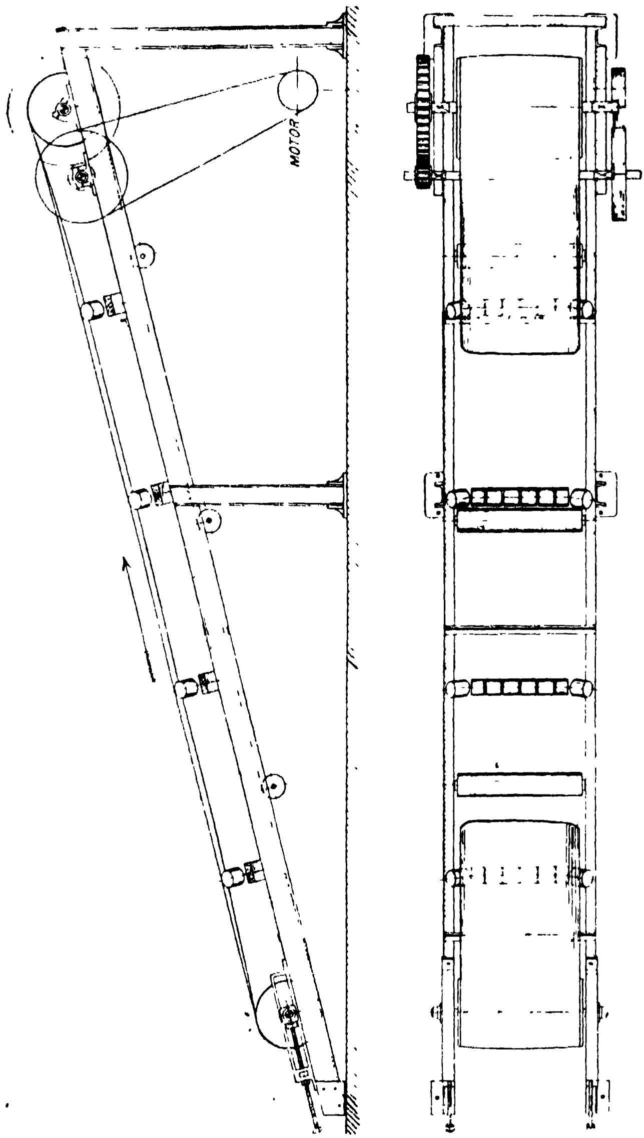

FIG. 251.—Elevation and Plan of Belt Conveyor.

and an elevation and plan of the same conveyor in Fig. 251. This conveyor is suitable for conveying large quantities of coal, for example, the raw coal to a washery. A smaller number of idlers would be required for smaller quantities.

The capacity of a belt conveyor is increased by increasing the inclination of the side rollers. The power consumption and wear of the belt increase more rapidly than the increase in the carrying capacity of the belt, and the inclination is therefore usually limited to 20° to 25°. Belt conveyors are loaded from other belts or from shoots which are inclined (in the direction of travel of the belt) to give the material loaded as nearly as possible the same speed as the belt. The load is usually discharged over the end driving pulleys at the end of the run, or they may be discharged at any point along the length of the conveyor by carrying the belt over a straight elevated pulley, and a lower straight pulley arranged slightly to the rear of the first. In the latter case the belt makes an S-loop.

The driving pulley is usually the one at the discharge end of the conveyor to keep the carrying portion of the belt in tension. The driving pulley should be heavily crowned to maintain alignment of the belt. Provision is made to take up extensions of the belt through stretching. Adjustment is usually made by moving the end pulley near the feed end. A belt conveyor may be inclined at angles up to, but not exceeding, 20° for dry coal. Rubber belts are expensive, but work without attention or vibration.

The capacity of a belt conveyor depends on whether the belt is troughed or is flat. A troughed belt carries twice the capacity of a flat belt of the same width. The capacity is given by the formula :—

$$Q_m = a\, v\, 60 \times 0{\cdot}8,$$

where Q_m = the capacity in metric tons per hour,
a = the area of cross-section of the coal on the conveyor in square metres,
v = the speed of the belt in metres per minute,
$0{\cdot}8$ = the density of crushed coal (water = 1).

The capacities of belts of different widths are given in Table 145,* in which a belt speed of 256 ft. per minute (1·3 m. per second) is assumed.

The power consumption is calculated from the formula :—

$$\text{H.P.} = x\,\text{L},$$

where x = a factor obtained from Table 145,
L = the length of the conveyor in feet.

B is a value to be added for each point of support of the belt ; z is a value to be added for short conveyors less than 20 m. (approximately 66 ft.).

When a conveyor is inclined the power consumption becomes :—

$$\text{H.P.} = x\text{L} \cos \alpha + \frac{\text{QH}}{900},$$

where α = the angle of inclination,
H = height in feet through which load is lifted.

* Adapted from Gröppel (*loc. cit.*).

TABLE 145.—CARRYING CAPACITIES AND POWER FACTORS OF TROUGHED BELTS OF DIFFERENT WIDTHS

Width of Belt.		Capacity (Q).	Power Factors.		
m.	in.	Tons per hour.	x	B (h.p.).	z (h.p.).
0·3	12	20	0·012	0·1	0·6
0·4	16	40	0·018	0·1	0·7
0·45	18	50	0·022	0·1	0·8
0·50	20	65	0·025	0·1	0·9
0·60	24	100	0·028	0·2	1·0
0·70	28	140	0·036	0·2	1·1
0·80	32	180	0·043	0 2	1·2
0·90	36	230	0·051	0·2	1·3
1·0	40	300	0·063	0·2	1·4

The factor $x\text{L} \cos a$ is the power used in transporting in a horizontal plane, and the factor $\frac{\text{QH}}{900}$ is introduced for the power absorbed in lifting. For example, a belt conveyor 100 ft. long, with six rollers, to carry 125 tons per hour, would be 0·7 m. wide, for which the power factor, x, is 0·036.

Power required $= 0·036 \times 100 + 6 \times 0·2 = 4·8$ h.p. If the same belt were inclined at an angle of 20°, that is it rose 34·2 ft. in 100 ft., the power required would be :—

$$4·8 \cos 20 + \frac{125·34}{900} = 4·5 + 4·7 = 9·2 \text{ h.p.}$$

Band conveyors may also be made of a series of steel plates, hinged normal to the direction of travel to enable them to pass over the driving drums. On the flat portion of their travel, one plate just overlaps the one in front of it. Similar conveyors for lump coal may be composed of bars, arranged in sections and spaced slightly apart. Sectional plate and bar conveyors are used as picking belts.

Jigging conveyors are used to convey coal for short distances in a horizontal direction or down a slight incline. Their surface is made of mild-steel plates, and they may be looked upon as jigging screens with the apertures closed. They are similarly constructed and similarly actuated. Their principal use is for transporting large quantities of coal, preferably nut coal, say, to the loading booms. They are made wider than other forms of conveyor, require less horse-power, and cause less fracture. They are, however, more

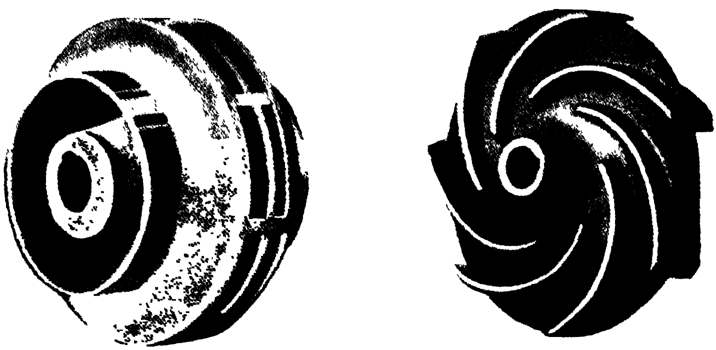

FIG. 253.—Views of Two Types of Impeller.

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expensive to instal unless the duty required is heavy. The ability of one unit to deal with large quantities then gives them an advantage over other conveyors.

PUMPS

Centrifugal pumps are almost invariably used for the circulation of water in washeries. A centrifugal pump consists of an impeller (which is a wheel carrying a number of suitably shaped vanes) enclosed in a casing. The centrifugal force due to the speed of the impeller gives the water in the casing a high velocity head, which is partly converted into pressure head (static pressure) as the

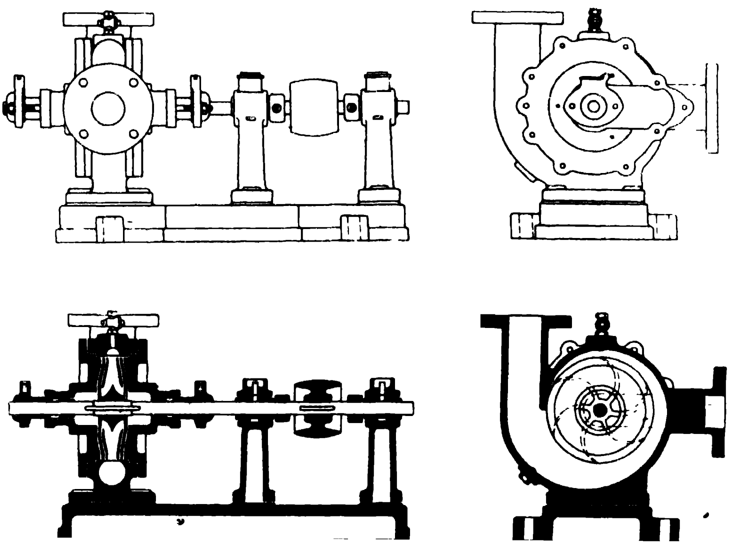


FIG. 252.—Elevation, End View and Sections of Centrifugal Pump.

water passes through the delivery pipe. As water is forced from the pump casing, fresh water is drawn in through the inlet pipe.

An elevation, end view, and sections of a centrifugal pump supplied to a Gröppel washery are given in Fig. 252. Two types of impellers are illustrated in Fig. 253. Type *a* is used for clear water or water containing a limited proportion of solid matter. Type *b* is used for slurry. The pumps are either belt driven or direct-coupled to a motor on the same bed plate. The capacity, power consumptions and other details of a typical centrifugal pump (the "Robusto," supplied by Fr. Gröppel, Bochum) are recorded in Table 146.

Thus, with a 12-in. pump, the power required to lift 2,530 gal. of water per minute to a height of 1 m. (3·28 ft.) is 6·8 h.p.; to raise the same quantity 2 m. the power required is (6·8 + 3·8) h.p.; for a 3 m. lift (6·8 + 2 × 3·8), and so on.

TABLE 146.—CAPACITY AND POWER CONSUMPTIONS OF CENTRIFUGAL PUMPS

Internal diameter of pipes (in.)	2⅜	3⅛	4	5	6	7	8	10	12	14
Hourly capacity gal. per min.*	73·4	147	257	404	587	826	1,065	1,725	2,530	3,580
Power consumption (h.p.)—										
(a) For 1 m. lift	0·6	0·75	1·1	1·3	1·5	2·0	2·3	4·0	6·8	9·5
(b) For each further 1 m. height of lift	0·15	0·28	0·44	0·67	0·96	1·33	1·64	2·62	3·80	5·12

* For water S.G. = 1·0.

One of the chief troubles experienced with centrifugal pumps is the wearing of the blades of the impellers and the gland packings. Fine particles of wet coal work their way into the glands, and once an entry is effected the packing soon becomes destroyed. Glands therefore require to be repacked at frequent intervals, and there is an evident need for some more resistant packing than is usually considered suitable for ordinary water pumps. The wear of the impeller blades is a frequent cause of washery pumps failing to give the required capacity. To improve the delivery, the pump may be driven more rapidly, and a belt drive enables this to be done more readily than if the pump is direct-coupled to the motor.

Some trouble is experienced with pumps to deal with slurry, on account of their liability to choke on standing. In some washers, the thickened slurry is run from the bottom of spitzkasten into a sump from which it is subsequently elevated by a centrifugal pump. The slurry is run out of the spitzkasten continuously whilst the washer is working and, with a continuous feed, the slurry pumps may work satisfactorily. When the pumps have been standing, difficulty may be experienced in restarting them because the slurry tends to settle into a compact mass. Slurry pumps are usually situated in a sunken pit, in which position they are apt to escape proper inspection and attention, although, with the heavy duty of handling a most abrasive material, they are liable to lead to much trouble. It is preferable to have a long pump suction pipe and to place the pump in a more accessible position. In the event of the suction pipe becoming choked it is then easier to dismantle or to clear it with water.

It may be noted, in passing, that the procedure of pumping the washery water to an elevated settling tank and running off the thickened slurry by gravity—as in the Baum washer, for example—is a much more rational practice than to use settling tanks at a low

elevation and to pump the thickened slurry to the dewatering screens.

The choking of centrifugal pumps is also a difficulty experienced in the disposal of effluent washery water. It is usually desirable to recover the coal particles which are suspended in the water, partly for reasons of economy, but also to render the effluent readily disposable. Such an effluent is obtained from drainage hoppers, from which the drainings may contain considerable quantites of coal. To avoid the difficulties associated with the choking of centrifugal pumps, an air lift may be used (*Coll. Eng.*, 1925, 2, 360). A compound air lift, as illustrated in Fig. 254, raises the effluent from the bottom of a 6 in. pipe, 6 ft. below the ground level, through a 3 in. pipe to a tank above ground level. This tank gives a constant feed to the second stage, in which the water is lifted through a 3 in. pipe to an overhead settling tank. In this case, compressed air was supplied at 10 to $12\frac{1}{2}$ lb. per sq. in. pressure, through two $\frac{3}{4}$ in. air pipes, the one in the first stage being in the centre of the lift pipe, and the one in the second stage being outside the lift pipe. This arrangement facilitates adjustment.

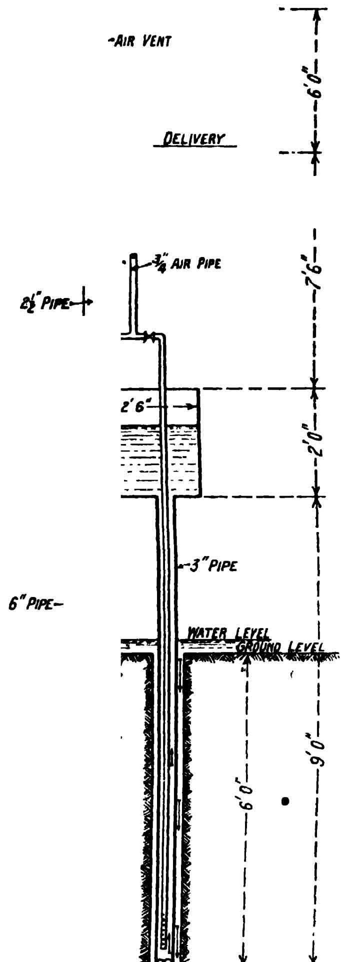


FIG. 254 — Air-lift for Disposal of Effluent Washery Water.

In practice it was found to be preferable to fix the aircock in a definite position; quantities varying from 2 to 20 gal. were lifted per minute with an expenditure of about $1\frac{1}{4}$ h.p. The lifter was left working continuously, and, when the washer was standing overnight, it served to lift the drainings without attention. No difficulty in choking of the pipes was experienced.

CHAPTER XXIX

SCREENING

General.—The screening of coal is a process complementary to the cleaning of coal, for, in some cases, the coal must be sized between certain limits before it is submitted to the cleaning process, and in others, screening is an accessory operation in the preparation of coal for the market. The sizing of coals for sale after cleaning, and when the coal does not require cleaning, is guided by convenience, local custom, and market requirements. In the anthracite market particular attention is paid to carefully sized products as an aid to efficiency in burning, and the sizes are chosen according to the ultimate use of the coal. In other markets, accurate grading is considered to be less important than in the anthracite market, and, indeed, there is no fixed scale of sizes by which bituminous coal may be classed. For this reason the nomenclature is somewhat confused. "Slack" in one district may differ considerably in size from slack in another district, and even the term "⅝-in. slack" is not specific, because the maximum linear dimensions of particles passing through a ⅝ in. screen will vary according to whether square or round holes, inclined or horizontal screens, are employed.

The Federation of British Industries has proposed to its members a standard table of size nomenclature as follows :—

No.	(a) Bituminous	(b) Welsh.	(c) Anthracite.
1	Large screened . . .	Large screened.	Large screened.
2	Large unscreened . .	Through-and-through	Through-and-through
3	Cobbles	—	Cobbles.
4	Treble nuts (say 3 to 1½ in.) .	Nuts.	—
5	Double nuts (say 1½ to 1 in.) .	Peas.	—
6	Single nuts . . .	—	Single Nuts.
7	Peas, beans or pearls * . .	—	Peas or beans.
8	Nutty slack (rough dross or small)	Small . .	Rubbly culm.
9	Fine slack (duff or dant) .	—	Culm.
10	Fine small (dross, dant, or duff)	Duff . .	—

* "Pearls" is said to be a registered trade name.

Even this is only a rough classification, and does not state the

sizes of the various grades. It would seem to be more desirable to define certain sizing limits as a basis of classification, or, alternatively, to insist that, when sized coal is marketed, the sizes of the screen apertures should be stated, whatever name be assigned to the grade. The difficulties would not be eliminated, but would be lessened if one colliery sold its nuts, for example, as trebles ($3\frac{1}{2}$ to $1\frac{3}{4}$ in.), doubles ($1\frac{3}{4}$ to 1 in.), singles (1 to $\frac{5}{8}$ in.), and smalls ($\frac{5}{8}$ in. to 0), and another as trebles (3 to 2 in.), doubles (2 to $1\frac{1}{4}$ in.), singles ($1\frac{1}{4}$ to $\frac{3}{4}$ in.), and smalls ($\frac{3}{4}$ in. to 0). The lack of uniformity in name would still exist, but the statement of the screen sizes would enable the buyer to differentiate more easily.

Complete uniformity is, at present, impossible, for collieries are equipped with screens having round holes, square holes, diamond-shaped holes, or slots, all of which give different maximum sizes of coal passing through the screen. Moreover, screening is never absolutely 100 per cent. efficient, and a given size of coal from one colliery may contain a greater amount of undersize than that from another colliery. To ensure the greatest uniformity, it would be necessary for all collieries to use screens with a given shape of aperture and to guarantee the coal to contain not more than a certain proportion of undersize. Even then, the delivered product would vary because of the different friabilities of coals.

The South Wales Coal Owners Association has recommended to its members a standard of sizing, and the sizes proposed are: No. 1, 80–55 mm.; No. 2, 55–25 mm.; No. 3, 25–15 mm.; No. 4, 15–8 mm.; No. 5, 8–4 mm.

It is not proposed to deal with the history of screening practice, but it is interesting to note that in 1740, nearly 200 years ago, " the mischievous practice of screening coals was first introduced at Willington Colliery by Mr. William Brown." (*Trans.* North of Eng. Inst. Min. Eng., 1865–6, 15, 205.) We shall confine our description to more modern and, we hope, less mischievous machines. It may be mentioned, however, that screening was fairly common in Durham in 1836 (*The Miner's Guide*, Thomas Smith, Sheffield, 1836). Screens, at first, were made of wicker work, and the bar screen was introduced by Hall in 1833. In 1844 a simple revolving screen was patented by Walker, and Bérard fitted screens in his jig washeries in about 1850.

Where there is no call for accurate grading, some of the earliest types of screen, for example, the gravity bar screen or grizzly, are still in operation. In general, however, modern screening plants are equipped with revolving screens, shaking screens, or vibrating screens, and the apertures in the surface are either round holes or square holes. Each of these four types of screen has its own particular advantages, and the choice of the best type for any purpose depends upon the circumstances.

Whichever type be considered the most suitable, there are certain qualities that all screens must possess if they are to be satis-

factory in use. The chief requisite of a screening plant is that it shall require practically no attention. It must combine the greatest possible simplicity with adequate strength and a reasonable accuracy. No screen is perfectly accurate, but the greatest satisfaction and accuracy is obtained if there is little breakage of the material on the screen, when the speed of travel of the material is uniform across its surface, and when there is little tendency for the screen apertures to clog or " blind."

Certain factors which affect screening are dependent upon the material treated. Pieces of coal and shale, for example, are of all and assorted shapes. A thin flat piece of shale may refuse to pass through the apertures in a given screen, whereas a cubical piece of coal of greater mass and greater width and thickness may pass through quite easily. The moisture content of the material will also affect the efficiency of screening. Other factors which influence the accuracy are the shape of the apertures, the slope and motion of the screen, and the size and velocity of the material. The greatest accuracy can be achieved when dealing with large, dry particles on a screen which has a vibratory or jigging motion, and which is nearly horizontal.

It is advisable, at the outset, to differentiate between screening as part of the routine of colliery practice, and screening as an adjunct to a washing plant. At many collieries the screening plant is considered to include gantries, tipplers, screens and picking belts. The screen in a washery is essentially a sieve or riddle, whose only purpose is to deliver certain material in fractions of different sizes. The colliery screening plant is usually part of the integral lay-out of the colliery's surface equipment, and some of the coal passes from the screening plant to the washery. The washery screen is in the washery building, in the most suitable position, and whereas the main screening plant can be equipped with the best types of plant, the washery screen must sometimes be of the type which will fit into a particular and limited space. Washery screening and pit screening are therefore different branches of the general problem and different factors are concerned in the two.

Colliery Surface Equipment between the Winding Shaft and the Screens.—In the lay-out of the surface plant of a colliery, the design of the screening plant (including gantries, tipplers and picking belts) depends largely upon the slope of the ground, the position of the shafts, the area and shape of the space available. Certain features, however, must be provided; for example, provision must be made to allow the trams to gravitate when pushed out of the cage and, with double-decking, to come to a common level at the weighing platform. Furthermore, the coal must not be tipped nearer than 80 ft. from the downcast shaft, and gravitation must be used, or creepers must be provided to remove the trams to this distance.

Creepers.—Once the trams have been weighed, the coal can be transferred to the screens by any convenient means. As a general rule, it is most convenient to convey the coal in the trams to tipplers directly above the screens. For this purpose the trams must travel along an inclined creeper, so that after leaving the tippler they will gravitate back to the top of the shaft. Alternatively, the coal may be tipped after the trams are weighed and be conveyed to the screens by belt conveyors. In general, creepers have an advantage over belt conveyors for the following reasons: (*a*) The upkeep of belt conveyors is heavy, (*b*) any stoppage may be serious, (*c*) the angle of inclination is limited by the tendency of particles to roll down the conveyor, (*d*) at least two tippings (from trams to the conveyor and from the conveyor to the screens) are necessary, with an increased tendency for breakage, (*e*) the length of a belt conveyor is limited by the necessity to interpose driving drums. Creepers, on the other hand, are cheap to maintain, stoppages are seldom serious, one tippage only is required, and the inclination is limited only by the tendency for the trams to tip up and for coal to spill over the sides.

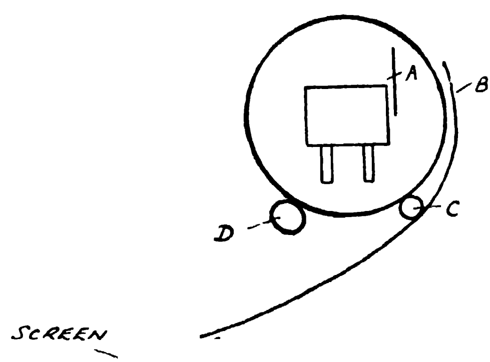


FIG. 255—Tippler.

Creepers have the additional advantage that they can be placed in the most convenient position and can allow the trams to remain under mechanical control as long as possible. It is inadvisable to have long lengths over which the trams run by gravity, because of the great differences between trams. When using gravity, the inclination must be great enough to eliminate the risk of stoppage with a badly running tram, and trams which run more freely than others acquire too high a speed. The use of creepers overcomes the irregularity which exists between different trams.

A suitable speed for creepers moving tubs uphill is 50 ft. per minute, with horns every 5 ft. A maximum of ten tubs can then be handled per minute, and, on the average, about eight tubs per minute will be conveyed.

Tipplers.—There are various designs of tippler available for discharging the coal from the trams to the screens. In all modern tipplers, the loaded tram entering the tippler knocks out an empty

tram, which, moving forward, trips a catch and causes the circumferential surface of the tippler to come into contact with a continuously revolving roller, C (Fig. 255). The tippler is then driven (in Fig. 255 in a clockwise direction), and the coal falls on to a plate, A, from which, later in the revolution, it falls on to a second plate, B. The plate, B, acts as a shoot to deliver the coal directly on to the screen, and the interposition of the plate, A, breaks the fall of the coal, and so reduces the amount of breakage. The roller, C, is power driven, and on completion of the revolution, the tippler is withdrawn from the roller, C, but not from a second roller, D, which is idle. From the tippler the coal passes down the shoot, B, to the surface of the screen below it. In general, a tippler cannot deal with more than seven trams per minute, and its speed of revolution is equivalent to 10 r.p.m. maximum during revolution.

Speedy operation of the tippler is a most important consideration. Otherwise, a stoppage occurring at the tippler, the track from the pit-top may become filled with loaded tubs, and if these cannot be cleared, winding must be stopped. The tippler must be able to work faster than the coal is hoisted up the shaft, so that, when the stoppage is cleared, the track can be emptied quickly and room be made to provide space for a subsequent stoppage. Usually, between the winding-shaft and the tippler, a stoppage of five minutes can be expected every hour, and the gantries should provide storage space for a ten-minute hoist.

Screens.—When a colliery is furnished with a washery, it is usual for the screens to remove the particles below, say, 3½ in., and these are passed forward along belt conveyors to the washer, or are loaded into wagons and discharged at the washery into an underground hopper at the foot of the raw-coal elevator. The lumps passing over the screens are hand-picked and loaded into wagons at the end of the picking belts or picking tables.

Practically the only type of screen employed to remove the smaller sizes for washing or prior to hand picking the lumps is the inclined jigging type, suspended by arms and actuated by an eccentric or crankshaft. The barrel type of screen has been discarded for a variety of reasons, chiefly because of its low capacity, and the amount of breakage which is caused. Barrel screens are cumbersome and trams of coal cannot easily be loaded into them, whilst the oversize delivered is not evenly spread out, as is required on a picking belt. Moreover, they tend to "blind" badly, especially on the outer mesh, and if the coal is damp. By comparison, inclined jigging screens are cheaper, have a high capacity, and give a well-spread discharge. They require, however, rather more horse-power.

There is little essential difference between the screens employed in a colliery screening plant and in a washery except that different types are sometimes more suitable for one than for the other. Whereas the inclined jigging screen is usually the best type for the

FIG. 256.—View of Picking Belts.

[To face page 541.

main colliery screen, the revolving barrel type is frequently preferred in a washery because it requires a small floor space and its mechanism is free from vibration. When, as frequently happens, the screens are near to the top of the washery building, the absence of vibration is an important consideration. Moreover, the coal is usually accompanied by a large bulk of water, and water is often sprayed on to the screen. In these circumstances, the capacity of a barrel screen is greatly increased and the tendency for it to blind is reduced.

Picking Belts.—Picking belts may be composed of an endless canvas or rubber belt, a wire mesh or steel plate surface, or may be made of a series of bars spaced a short distance apart. Endless belts of any of these types are driven by a driving drum at the feed end of the belt, and pass over a second drum (usually idle) at the discharge end, thus being in continuous circulation. The return journey of the belt is always made underneath the picking surface. Bar belts enable fines made during picking to fall through the gaps and to be separated from the lumps. The fines are then scraped back on the return journey of the belt by angle-scrapers fixed to its surface, and are mixed with the small coal previously removed by screening. A belt composed of plates is, however, cheaper both in initial cost and in upkeep, and causes less breakage as the coal falls on to it from the screens, or off it at the loading end. Moreover, it is easier to pick on a flat smooth surface than on an uneven surface composed of bars. A view of a picking-belt installation is given in Fig. 256.

The chief requirements of a picking belt are freedom from wear, security of the plates (or bars), and easy working. This is achieved by accurate alignment of the faces of the driving drums, uniformity of the connecting links, and accuracy of the link-centres (which are usually about 7 in. apart).

Picking belts are also conveyors and may be duplicated, one belt acting as a picking table for the lumps, and the other one, parallel to it, as a conveyor for the smaller sizes; both grades of coal can then be loaded into wagons at the same position.

The capacity of picking belts varies according to the nature of the raw coal, but at South Yorkshire pits, 100 ft. length of belt is usually sufficient and six boys are required for an output of 1,000 tons per shift.

S. R. and W. H. Berrisford (*Trans.* Inst. Min. Eng., 1924–25, 69, 282) give the capacity of picking and conveying belts as follows :—

Duty.	Length. Ft	Width Ft. in.	Speed. Ft per min	Capacity Tons per hour.	H.P. required.
Slack conveying . .	130	3 6	53	150	1·75
Main coal belt . .	250	4 6	56	150	7·25
Coal picking . .	55	4 6	50	350	6·0
Do. (four belts) .	—	4 6	60	200	8·0

SCREENS

As already stated, there are four main types of screen suitable for colliery use, namely :—

(*a*) Gravity screens.
(*b*) Revolving screens.
(*c*) Shaking or jigging screens.
(*d*) Vibrating screens.

These may be described separately.

Gravity Screens.—Gravity screens are always stationary, the coal travelling down them under the influence of gravity alone. They must therefore be fixed at an angle to the horizontal so that the material to be screened will slide freely from one end to the other without the aid of external forces. The angle of inclination necessary for this purpose will vary for a number of reasons, and in deciding the angle for any particular coal-screening plant it is necessary to bear in mind the following points :—

(*a*) Coal from various seams and different qualities of coal and dirt may pass over the screen and the angle must be set to accommodate the material with the highest coefficient of friction.

(*b*) There may be a stoppage on the screen, and it is desirable that the angle should be in accordance with the coefficient of static friction so that the material will begin to slide when the stoppage is removed.

(*c*) Allowance must be made for the effects of a rusty surface and of excessive moisture in the coal.

(*d*) The greater the angle, the greater is the headroom and power required for elevation, and the greater will be the tendency of the particles to roll down the screen and escape proper sizing.

(*e*) When the screen has once been installed it usually constitutes an integral step in the flow sheet, and the inclination cannot often be varied without necessitating alterations to the heights of shoots or elevators.

Some information with respect to the angles at which particles of coal and dirt begin or continue to slide on various surfaces was given in Chapter XIX in connection with dry-cleaning process dependent upon the different coefficients of friction of coal and shale. Coals from different sources have different coefficients of friction, and the numerical value of the coefficient of friction of any one particular coal will vary according to its size and moisture-content. The difficulty of designing gravity screens for meeting all the conditions at a pithead is, therefore, apparent. The slope must not be too small, or stoppage will result; nor must it be too great, or there will be considerable fracture of the coal and the sizing will be unsatisfactory.

The commonest form of stationary gravity screen is the bar

screen. The bars are fixed longitudinally down the screen, and the spaces between them are long and narrow. Quite large thin particles may therefore pass between the bars when it is not desired that they should do so.

The types of bar employed are various. Eleven separate types are shown in Fig. 257. All but No. 1 introduce a tapering effect which tends to reduce blockage of the bars and thus to make available the maximum screening area. Bars of this type also tend to reduce the breakage resulting from chipping of particles which have stuck in the apertures. Nos. 2, 3, 4, 9 and 11 are designed to enable small particles to be removed rapidly, the greatest width, in section, being below the top of the bars. Nos. 5, 6 and 10 are designed to use the minimum weight for bars of the flat type. There are certain advantages in having a flat screen surface, but for most purposes it is an advantage to reduce the area over which coal is in contact with metal. This is accomplished in Nos. 4, 8, 9 and 10 by means of a rounded top of the bars. The lumps then ride on a narrow surface, and it is said that they meet less frictional resistance. With Nos. 2, 3 and 11, and to a less extent with Nos. 4 and 9, there is a tendency

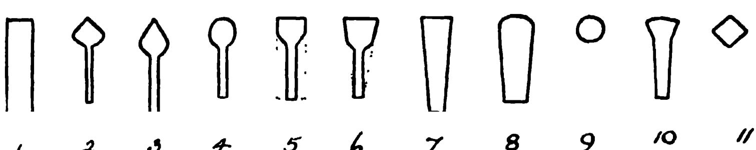


FIG. 257.—Sections through Bars used in Gravity Screens.

for the lumps to tip over and release fine particles which may be adhering to them.

The bars of the screens are held together between distance pieces by rods passing through holes in each bar, or by cross-bearing bars, notched to receive them. The cross bars are usually about 3 ft. apart.

In practice, gravity bar screens are most useful for removing nuts and smaller sizes from lumps before the lumps pass forward to picking belts. The raw coal is usually delivered to the bars by a shoot, which reduces the velocity of the coal particles, and the screened coal is discharged on to shoots with a greater angle of slope, to ensure rapid clearance. Usually gravity screens are 3 to 6 ft. wide, and 6 to 20 ft. long, and are inclined to the horizontal at inclinations varying from $4\frac{1}{2}$ to $7\frac{1}{2}$ in. per foot, the slope being greater, the smaller the coal. Statements of capacity of a gravity bar screen are very misleading, for the capacity depends essentially upon the relation between its length and width, the size and character of the material fed, the size of the material to be removed, and, especially, upon the regularity of the feed. About 80 to 100 tons per hour may be taken as a rough value for the capacity of a screen 6 ft. wide removing particles below $1\frac{1}{2}$ in.

Gravity bar screens are cheap to instal and to operate, and the maintenance cost is negligible. Unfortunately, however, they are inefficient, effecting only a very incomplete separation of small from large coal; they result in considerable breakage, and require considerable headroom.

In other forms of gravity screen, a perforated plate or a wire mesh is used for the screening surface. Such surfaces prevent the passage through the screens of large flat pieces, and much smaller sizes can be dealt with than when bar screens are used. The apertures are round, square, or rectangular. When they are rectangular, the longest side of the aperture is usually parallel to the direction of the material passing over the screen, but occasionally inclined apertures are used, the perforations in the surface being inclined both to the length and width of the screen. Rectangular perforations are also frequently staggered. Whether the screening surface is composed of bars, wire mesh, or perforated plate, however, the inefficiency of stationary gravity screens, and the headroom required, render them of limited applicability.

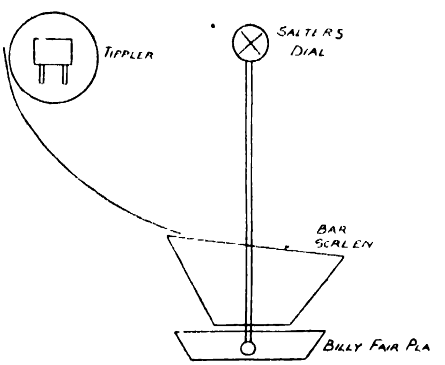


FIG. 258.—Billy-fair-play System.

Another form of stationary screen is the drag-screen, on which the particles do not move by gravity, but are moved along the surface by scrapers. As a rule the particles move down the screen, but drag-screens have been used on which the coal is scraped up an inclined surface. Screens of this type are but little used for sizing coal because of the great amount of breakage which results, but they are becoming popular for removing excess water from washed coal. In this form they are known as drainage conveyors, and the coal usually travels up a stationary inclined surface composed of bronze wedge wire, the water passing through the apertures.

Grizzlies are commonly found in South Wales as part of the Billy-fair-play system, in which they are used to separate the fines from the coal sent to the surface by the miners. The Billy-fair-play is a receptacle under the screen, which collects the smalls for weighing. The arrangement is shown in Fig. 258.

The loaded pit tub is tipped over the screen, on which the bars are $1\frac{1}{4}$ in. apart, and which is inclined at an inclination of $4\frac{1}{2}$ in. per foot; the smalls fall through into the receptacle below and the weight is indicated on a Salter's dial, from which the receptacle is suspended.

For purposes of this type the gravity bar screen is a useful appli-

ance in colliery practice. It is also quite satisfactory for coke screening, for coke is less easily fractured than coal, and is usually dry. In the removal of breeze from the large coke, great accuracy can be sacrificed for low cost and fairly high capacity. The efficiency of all such screens is increased by introducing lips or steps over which the coal cascades. The lip screen is popular in America.

Another type of bar screen is the moving bar screen, which is more efficient than the stationary bar screen, and requires less head-room, but which results in greater breakage of a friable coal. Alternate bars are made to move upwards and downwards, and also forwards and backwards by means of eccentrics. In other forms of

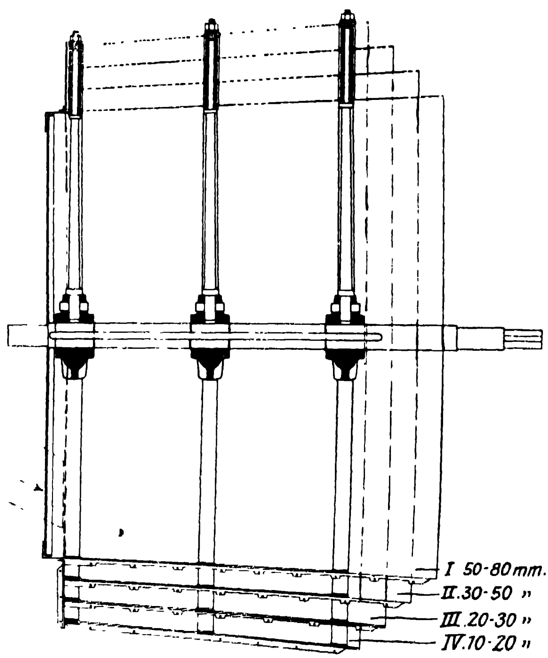


FIG. 259.—The Trommel Screen.

moving bar screen one set of bars is actuated by one eccentric and alternate bars are actuated by a second, set at 180 degrees to the first. An approximate figure for the capacity of such screens is about 100 tons per hour for a screen 6 ft. wide, inclined at about 2 in. per foot, with each eccentric making a stroke of about 4 in. at 50 r.p.m.

Revolving Screens.—Revolving screens or trommels were at one time the commonest form of screen found in coal washeries. They enable a fairly accurate gradation of the coal to be accomplished in a smaller space than with stationary screens, and at little cost. They have the disadvantage, however, that considerable breakage

of the coal results. A section through a trommel is shown in Fig. 259.

The trommel is composed of steel screen plates attached by spiders to a rigid cylindrical framework mounted on a central shaft. The shaft is supported on rollers at a suitable inclination to the horizontal, and is rotated by gearing at the lower or discharge end. In another form of trommel, the screening plates are bolted to longitudinal tee bars, outside the shell, and further strengthened by circumferential bands. Tyres are secured to the ends of the cylinder and are supported on rollers keyed to positively driven shafts. In this form of trommel, the drive is by bevel gearing at the upper end. The two forms are shown in Fig. 260.

The trommel may consist of several concentric cylindrical screens, each cylinder having a different size of perforation to enable a

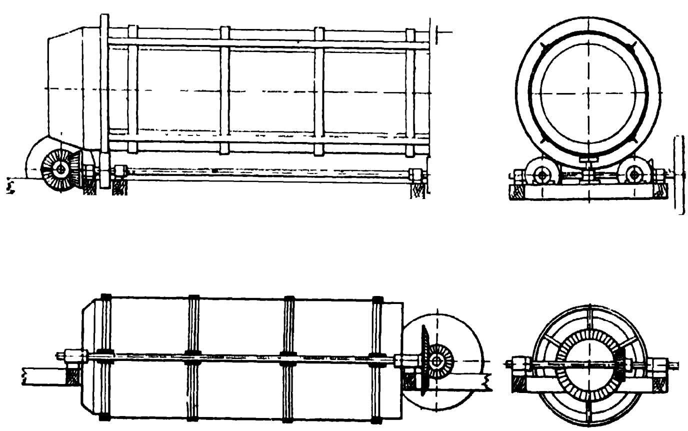


FIG. 260.—Cylindrical Screens : Diagrams.

number of products to be made. Alternatively, it may consist of a single cylinder, making two products only, an oversize and an undersize. Occasionally, the screen will consist of a single cylinder with different screening sections, the feed end of the screen having perforations of a certain size and the delivery end perforations of a larger size ; there may, in addition, be an intermediate section with an intermediate size of hole.

The ordinary cylindrical revolving screen is fixed with its axis slightly inclined to the horizontal. In general, for coal screening, an inclination of about 1 in 12, or 5 degrees, is satisfactory. As the screen revolves, the force of friction keeps the particles in contact with the screen as the surface moves upwards. When the coal is at the lowest point on the periphery of the screen, the friction between the coal and the surface is at a maximum. The higher the point on the periphery of the screen to which the particles are dragged, the less is the influence of frictional forces and the greater is the tendency

of the particles to slide down the side of the screen. The frictional force is supplemented, however, by the centrifugal force acquired by the particles, which tends to keep them in contact with the surface. For a given speed of rotation the centrifugal force is constant, and the gravitational forces eventually overcome the frictional and centrifugal forces, and the particles slide down the side of the screen. Because the screen is inclined, the position to which they slide is nearer the lower or discharge end, and the coal gradually makes its way through the barrel of the screen.

When the material is fed in bulk, a thin bed or bank of material is formed, rising up on one side of the trommel. If the speed of revolution is too slow, the centrifugal force acquired by the particles is small and they do not mount up the side sufficiently, leaving a bed too thick to allow all the particles a sufficient opportunity to fall through the apertures. If, on the other hand, the speed of revolution is too great, the centrifugal force is so high that the particles may tend to stick in the apertures. It is thus apparent that the centrifugal force of the particles is an important factor in the operation of revolving screens.

The rate of revolution chosen varies according to the size of the screen and of the coal. Small coal, for example, requires a higher rate of revolution than large coal. Screens of large diameter are revolved more slowly than smaller screens in order that the peripheral speeds may be more nearly the same. To obtain the highest capacity with nut sizes of coal, peripheral speeds of about 200 to 220 ft. per minute are used in America, but in England it is the practice to work with peripheral speeds rather lower than this maximum.

The capacity of revolving screens depends upon the speed of their revolution and their angle of inclination. The greater the angle, the more rapidly does the material pass over the screen, and, whereas the capacity is greater, the opportunity for the undersize to pass through the apertures is less. In general, when there is a small quantity of undersize, the amount of inclination may be more than 5 or 6 degrees, but, if it is much more than this, say 8 or 10 degrees, the thrust on the bearing at the lower end of the central shaft is excessive and may give rise to mechanical troubles.

The capacity of all screens depends essentially on the relative proportions of oversize and undersize, but it also depends upon the size and spacing of the holes, the dimensions of the screen and the regularity of the feed. About 4 to 5 sq. ft. of screening area in a revolving screen is usually required per ton of coal per hour with holes 1 in. in diameter, but about half this area is sufficient with 2 in. holes. Thus a screen 6 ft. in diameter and 10 ft. long with 1 in. holes, can usually handle about 60 tons per hour of coal. This figure assumes a regular rate of feed at a maximum of about 1 ton per minute. If the feed is periodically below the maximum the output suffers, for the leeway cannot be made up by heavy feeding.

The power required varies according to the amount of coal put over the screen, for at each revolution a certain amount of coal is lifted up the side of the screen. Assuming a regular load, the horse-power required is about one-tenth of the hourly capacity in tons. Thus a plant screening 60 tons per hour requires about 6 h.p.

The disadvantages of trommels are that they cause considerable fracture of the raw material (mainly as a result of collision with the arms), and give rise to considerable mechanical difficulties in driving, because of their inclination, which exerts an enormous thrust on the lower end bearing. They are also inefficient in dealing with fine coal (below, say, $\frac{1}{2}$ in.). On the other hand, with nut sizes they have a fairly high efficiency, for the coal particles are turned over and over and have many opportunities of passing through the apertures. Moreover, they work without vibration (and can therefore be placed at the top of a building), require little attention, and wear satisfactorily. The efficiency is greater with dry particles than with moist particles, for in the latter case the finest pieces tend to stick to the surfaces of the larger, and all the particles tend to slide over the screening surface, whereas, when they are dry, there is a greater tendency to roll and screening is then more efficient. The greatest efficiency is obtained, however, if the particles are sprayed with water at a high pressure whilst they are being screened. In these circumstances, or when the screening is accomplished in a stream of water, the water not only separates the particles from each other, but it tends to drag them through the apertures, so that, with water spraying, trommels can be used for the small sizes of coal down to about $\frac{1}{32}$ in.

Revolving screens are occasionally employed in cement mills to remove particles which have escaped grinding. This rather suggests that they are efficient with very fine particles, but powders behave differently, in screening, to material such as coal, of assorted shapes and sizes.

A modification and improvement of the standard type of trommel is the use of a revolving truncated cone instead of a cylinder. By this means the end thrust on the lower bearing, already mentioned, is avoided and the drive is communicated by a horizontal shaft. This advantage is partly outweighed, however, by the fact that the heaviest load is borne where the cone has the smallest diameter, and because the peripheral speed is unequal at different positions along the length of the screen. If the screen is used to separate the coal into several sizes, the peripheral speed is least, the bulk of material is greatest, and the bed thickest, where the smallest size is removed. Each of these three conditions is adverse to the most satisfactory operation.

The last two of these defects are common to all revolving screens which make several sizes by using a single shell, whether it is cylindrical or conical in shape. The defect is simply magnified by the lower peripheral speed of the feed end when the screen is cone-

shaped. When a single jacket is employed, the smallest sizes must be removed first and the largest sizes subsequently. This order is opposed to the greatest efficiency, for, whereas fine particles require a greater screening area than large particles, the bulk of the material is greatest in the first portion of the screen. Another disadvantage is that large particles, which fracture more easily than small particles, are retained for the longest time in the screen and are subjected to considerable breakage.

These defects are overcome in revolving screens consisting of two or more jackets. In these circumstances, the inner jacket contains the largest apertures, and the largest particles are separated first. The small particles can then spread themselves into a thinner layer in the outer jackets with a better chance of being properly sized. For this reason, multiple jacket screens, whether cylindrical or conical in shape, are more efficient than screens consisting of a single cylinder (or truncated cone) when it is desired to make several sizes in one operation. Moreover, the disadvantage suffered by conical-shaped screens, that the peripheral speed varies along its length, becomes of less importance because the smallest particles are separated in the outermost jacket, in which the peripheral speed is relatively the greatest throughout the length.

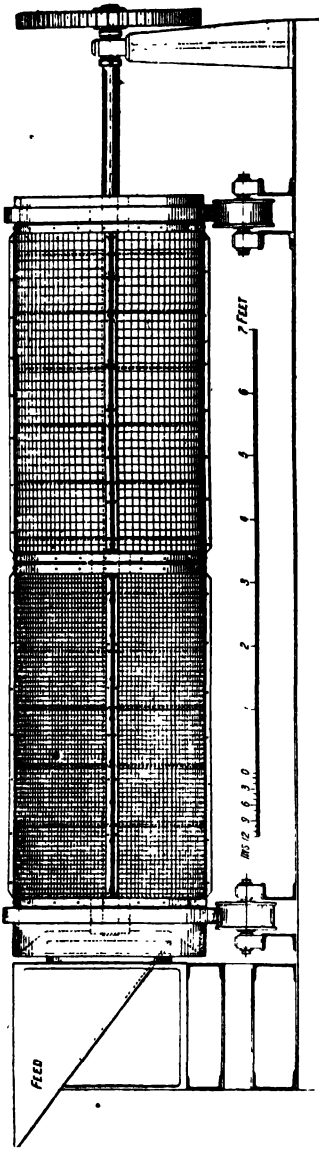
Although multiple jacket screens have these theoretical advantages, screens of a single cylinder, containing sections with different sizes of apertures, are sometimes used in preference to them. The principal reasons for this are that, to repair or replace the inner screens, the outer ones must first be removed, and that multiple jacket screens have, of necessity, a large diameter and the weight cannot be distributed over a long length. Moreover, with a very friable coal, a greater amount of fracture may be suffered as the particles fall from one jacket to another.

The truncated cone trommel with multiple jackets is probably the most suitable screen for general washery practice, because of its freedom from mechanical trouble and the relatively small floor space required. For wet material, sprayed with water during screening, the jackets are made parallel and inclined at an angle of 3 degrees to the horizontal and central shaft. With dry material, an angle of 8 degrees is required. Two disadvantages of the multiple jacket, conical-shaped screen not previously mentioned, are that some of the particles are passed through the apertures near to the discharge end of the inner jacket and there is scarcely any area for them to be screened on the outer jackets; and, secondly, that a large area of screening surface must be carried at the discharge end of the screen where the bulk and weight of material is smallest.

In German practice, the Schmidt screen, consisting of a single plate in the form of a spiral, was at one time extensively used. The material was fed to the core of the spiral and passed from the centre to the outermost shell, being completely screened in one revolution.

The shells had different sizes of apertures, and the sizes were kept separate by partitions in the annular spaces.

Another modification of the usual type of rotating screen is the use of polygonal screens instead of cylindrical screens. The flat plates of the polygonal type help to break up an agglomerating mass and are more easily replaced if worn. They cause more fracture than cylindrical screens, however, and are therefore unsuitable for coal.



Square-mesh woven wire sections are often used instead of perforated plates in all types of revolving screen. A far greater proportion of the total area of the screen then consists of apertures with a consequently increased capacity. Such screens, however, wear more rapidly than those with perforated plates.

The T. K. Screen.—A modification of the usual type of revolving screen is the T. K. screen (Messrs. R. H. Kirkup, Gateshead-on-Tyne), which consists of a woven-wire cylinder mounted horizontally. The cylinder is made up of one or more sections, each with a different size of aperture to enable different sizes of coal to be made. Instead of the coal travelling along the screen by gravity, however, it is caused to travel forward by metal plates inside the cylinder.

The screen is shown in Figs. 261 and 262. The wire-mesh surface of the cylinder is composed of steel wire built in sections upon a steel frame. There are four sections in the circumference, and, generally, the length is divided into two parts, as in Fig. 261, the first part removing the smallest material and the second part a larger size of material. The oversize is collected at the discharge end of the screen. The cylinder, at each end and at the division between the different mesh sizes of the screen, is fixed rigidly to the central shaft by spider castings or spokes. The faces of the spokes are set at an angle, so

that when the coal inside the cylinder comes into contact with them, it does not hang, but slides forward.

Each end of the cylinder is framed in an iron girth wheel or runner band, which is supported on iron rollers, and the cylinder can be revolved either by driving the rollers or by driving the central shaft. The latter method is preferable, as it places less strain on the mesh wire.

The central steel shaft is shown diagrammatically in Fig. 262. It is made of mild steel and has mild steel plates attached to it between distance pieces. Each of these plates is flat for a portion occupying three-fourths of the cross-section of the cylinder and normal to the central shaft. The remaining one-quarter of the cross-section is occupied by an inclined portion, which connects one plate with the one next to it. The plates divide the cylinder into compartments, and the inclined portions of them act as scoops, and cause the coal, after it has remained in one compartment for three-fourths of a revolution, to be moved forward to the next compartment. The inner shaft, with the plates fastened to it, and the screen mesh, all rotate together. Suppose that coal is in the first compartment; it will, of course, be resting on the wire mesh, and as the cylinder rotates, will be screened. After three-fourths of a revolution the inclined scoop gathers it up and sweeps it upwards with it. As the scoop rises, the coal slides off it and into the next compartment. By this means it gradually traverses the whole screen. It would be noticed that this device does not act as an ordinary screw conveyor, in which the screw revolves and the outer cylinder is stationary. In the T. K. screen the cylinder revolves with the screw.

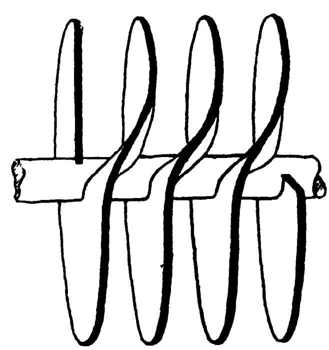


FIG. 262.—Screw of T. K. Screen: Diagram.

In Fig. 262 the joints between the straight ($\frac{3}{4}$) plate and the inclined scoop ($\frac{1}{4}$) are not shown, and the distance pieces and collars fastening the plates to the shaft are also omitted. In operation the feed is continuous, and each compartment contains coal in various stages of screening.

The capacity for an average British coal is approximately as follows:—

Diameter of Cylinder. Ft.	Capacity with Apertures of 1 in. sq. Tons per hour.
$2\frac{1}{2}$	8
3	15—20
4	20—40
5	40—90

The lengths of cylinder vary according to the proportion of undersize, and the horse-power required varies from 2½ to 10 h.p., according to the length of the screen. When the screen is composed of two sections of different aperture size, the scoops forming one-fourth of each plate of the conveying screen are arranged in line in each section, but the scoops in one section are placed at 180 degrees to the scoops in the other section for balancing purposes.

The chief advantage of the T. K. screen over inclined cylindrical screens lies in the fact that the driving shaft is horizontal, but another advantage is that the plates of the screw device prevent particles from rolling over the top of the bed and escaping screening.

The chief disadvantage would appear to be that the wire-mesh surface is worn by abrasion more rapidly than perforated plate.

Shaking or Jigging Screens.—Shaking or jigging screens are rapidly displacing trommels in colliery practice. They are more accurate, more accessible, cheaper to operate, and have a considerably greater capacity. Where possible, they are used in washeries, but for washery practice the vibration and the length of screen are often serious objections.

A shaking screen consists of a flat perforated surface mounted in a framework and inclined slightly to the horizontal. The framework is connected to an eccentric, or other driving mechanism, which can cause a reciprocating motion, and material fed to the upper end of the screen travels down it at each backward stroke, when the supporting surface is, as it were, suddenly withdrawn.

For colliery practice, shaking screens have the advantage that they can also serve as picking tables and that they act as conveyors and enable the products to be discharged directly into hoppers or wagons at different points. The screen can, indeed, be converted into a conveyor by "veiling" or covering it by a flat plate. They are often mounted side by side or one above the other, and they are also frequently hung in pairs so that their to and fro motion does not shake the building unduly. In these circumstances they can be balanced by using two eccentrics, 180 degrees apart, on the same driving shaft.

Generally the screen consists of perforated steel plates bolted to longitudinal steel channels or angles, and supported by cross pieces beneath the surface. This gives a rigid structure capable of withstanding heavy service. For screening the smaller sizes a lighter framework may be used. The American practice of using a wooden framework finds little favour in this country, and even for light work a steel frame is usually adopted.

The screening surface consists of punched steel plate, or, for the smallest sizes, wire-mesh gauze. Gauze will not lie flat, and requires more support than plate, the supported parts being blinded. The wear is also greater, and, because of the greater frictional resistance, a greater slope and more headroom are required.

Perforated plates are cheaper and have a longer life, and are also easier to renew.

The use of jigging screens for dewatering washed fine coal is rapidly establishing itself in England, especially in those systems employing Baum washers. For the purpose of dewatering, the screen is composed of wedge wire rods arranged longitudinally. The rods are triangular in section, and one of the flat sides of a series of rods forms the surface along which the material slides. The water passes through the gaps between the rods (carrying with it perhaps some of the finest particles in suspension), and is rapidly removed from the expanding openings below the surface.

The mechanism for causing the jigging motion of the screen consists, usually, of a simple eccentric and driving rod, giving a simple harmonic motion. The motion is varied somewhat by different makers. In some types, the drive is communicated through a crank shaft which also produces a regular reciprocating motion. In other types, the forward stroke is one of rapid acceleration, changing instantaneously to one of rapid retardation during the backward stroke. This motion is familiar in the design of concentrating tables, and its object is to facilitate the passage of particles from one end of the surface to the other. With a motion of this type the screen can be erected more nearly horizontal than a screen actuated by an eccentric or a crank shaft, and the crisper movement helps to throw particles out of apertures in which they may tend to stick.

Most jigging screens are arranged so that during the forward stroke the motion carries the screening surface slightly upwards, and slightly downwards during the backward stroke. This again facilitates the passage of the material over the screen. The principle was early employed in the Ferraris screen, or zimmer, which consists of a horizontal tray with a perforated base, supported on inclined flexible legs. The motion imparted by an eccentric causes alternately a forward and rising motion and a backward and downward motion. This arrangement, though admirable for the purpose of conveying the material from one end of the screen to the other, is not always altogether satisfactory from the screening point of view, for the material tends to travel forwards with a series of hops and is not constantly in intimate contact with the screening surface.

In the ordinary inclined type of jigging screen the speed and stroke of the eccentric and the angle of inclination of the screen are related. These variables must be so arranged that when the screen is withdrawn at the end of the forward stroke of the eccentric the particles must be able to slide down the surface, the rate of withdrawal being such as to overcome the force of friction between the two surfaces in contact. During the forward stroke, the eccentric causes an acceleration of the screen surface, and if the acceleration is sufficiently rapid, there will be a tendency for the particles to move up the screen. This movement would tend to retard the progress of the particles over the screen and reduce its capacity.

There is, therefore, a maximum speed of acceleration which must not be exceeded, as well as the minimum rate of withdrawal of the screen, below which the particles will not slide down the screen on the backward stroke.

Holbrook and Frazer (U.S. Bureau of Mines, Bull. 234, have shown that these limiting conditions may be expressed as follows :—

$$\text{Maximum}: \quad \omega^2 R = \frac{(\mu \cos \alpha + \sin \alpha)g}{(\cos \alpha - \mu \sin \alpha)} \quad . \quad . \quad . \quad . \quad . \quad . \quad (45)$$

$$\text{Minimum}: \quad \omega^2 R = \frac{(\mu \cos \alpha - \sin \alpha)g}{(\cos \alpha + \mu \sin \alpha)} \quad . \quad . \quad . \quad . \quad . \quad . \quad (46)$$

in which ω is the angular velocity (or revolutions per minute of the eccentric) and R its radius (or half its stroke), μ is the coefficient of friction, and α the inclination of the screen. The equation (45) gives the maximum condition, and equation (46) the minimum condition. For a given slope of screen and a given coal, the right-hand sides of each equation become constant, and the limiting values of the multiple ω^2R may therefore be calculated for any known set of conditions.

Although these general conditions are of wide application, circumstances frequently warrant a value of ω^2R greater than the stated maximum. With small coal it is an advantage to have the particles moving slightly upwards when the screen moves forwards, for the bed is then kept loose and the particles are shaken about, so that the smaller sizes can reach the screen surface. It is also possible to exceed the maximum with large coal without decreasing the capacity, but practical limitations are imposed, because, if the multiple ω^2R is increased unduly, the vibration is excessive, close screening is sacrificed, and fracture of the coal is more severe.

The speed of passage of coal over inclined screens should, in general, be about 30 to 40 ft. per minute. Above this speed there is a tendency for breakage and the screening is not efficient.

The commonest form of jigging screen in use in Great Britain consists of a perforated tray suspended by hangers. The screen made by Plowright Bros. is inclined at various angles from the horizontal up to 15 degrees, and the hangers make angles with the vertical which vary from 15 degrees if the screen is horizontal, to 0 degrees if the screen is inclined at 15 degrees. There is thus always an angle of 75 degrees between the screen and the hangers in the mean position. For a screen fixed at 15 degrees to the horizontal, a stroke of 5 in. is usually imparted. With a horizontal screen the stroke is increased to 7 in. The speed of the jigging motion varies, but is greater when the stroke is shorter. A stroke of 5 in. and a rate of revolution of the eccentric of 100 r.p.m. is usually satisfactory for the main colliery screening plant. The screens are arranged in pairs, one following after the other, and may also be so arranged

that they consist of different decks, the undersize from the upper deck being screened on a lower deck.

When used for removing dry dust, of size less than, say, 1 mm., the screen consists of wedge wires arranged longitudinally and hung on hangers in the same manner. The stroke, however, is only 2 in. and the vibration has a frequency of 240 per minute. For coal which is damp, the ordinary screen may be actuated by a bumping cam underneath the screen, bearing on a block. In these circumstances also the inclination of the screen must be increased to 30 degrees.

The suspension of shaking screens is a matter of considerable importance. On poorly built screens, mechanical difficulties frequently arise owing to excessive wear in the driving and supporting mechanism. The troubles can often be traced to dirt getting into the eccentric drive, or to bad balancing of the drive, as a result of which there is an excessive and uneven wear on the suspension bearings and the screen develops a wobble.

To overcome defects in the suspension the screens may be

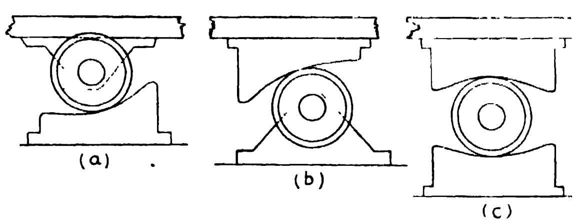


FIG. 263.—Roller Supports for Jigging Screens.

driven on rollers running on a track, and this method of support has been occasionally used in preference to hangers. Hung screens are, however, better than roller screens, provided that the screens are properly designed. With roller supports, the path of the screen can be made to suit the circumstances. The track may be horizontal, giving a motion very similar to vertical hangers, or may be inclined to give an upward motion on the forward stroke and a downward motion on the backward stroke. Curved tracks can also be used, giving a sharp reversal of direction at the end of each forward or backward stroke. All roller-supported screens require more power than suspended screens, and the vibration of the supporting structure is therefore greater.

Roller supports may be made so that the rollers are fixed to the screen plate (*a*, Fig. 263) and run in a fixed track on the supporting structure. With this arrangement dust collects on the bearing surface and greatly increases the rolling friction and wear. A second method of support (*c*, Fig. 263), open to the same objection, is to employ a loose roller bearing against two fixed surfaces, one attached to the screen and one to the supporting structure. This arrangement

has the advantage of a minimum bearing friction and has been extensively used in screening plants. A simpler method of construction (*b*, Fig. 263), which minimises the accumulation of dirt on the bearing surface, is to employ a fixed roller on the supporting mechanism and to attach the track to the screen. It is really an inversion of the first method.

Hung screens may be suspended on simple rods or wooden staves, but the best method of suspension is provided by a short-arm or drop-shaft hanger. Wooden hangers, as used in America (in the Parrish screen, for example), are 6 to 8 ft. long, and the screen there-

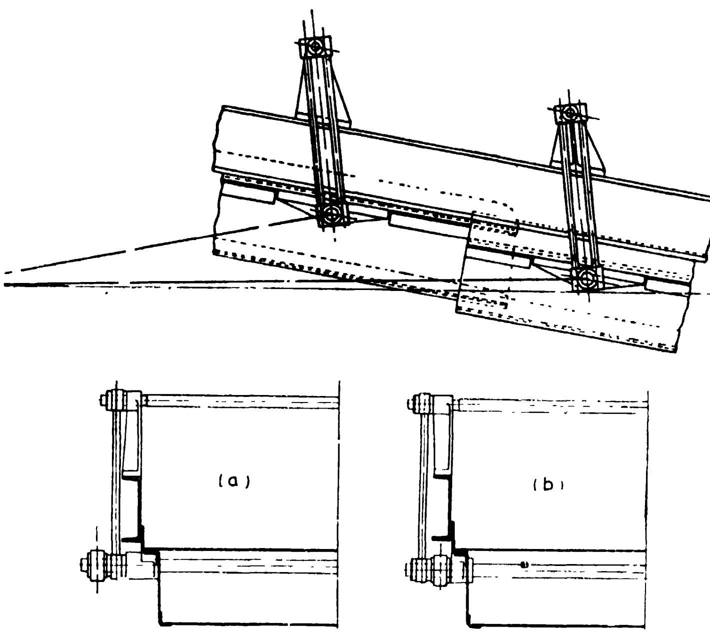


FIG. 264.—Hanger Supports for Jigging Screens: (a) Driver outside Hanger (b) Driver inside Hanger.

fore moves through about a 6 in. arc of a circle of 6 ft. radius. Its motion is, therefore, practically horizontal. The advantage of a forward upward motion in keeping the bed loose, maintaining a high capacity, and throwing particles out of apertures in which they may have stuck, are therefore lost. To obtain these advantages, short arms must be used for suspension, but the flexion on short wooden or metal hangers when they are rigidly attached at one end to the screen and at the other to the supporting structure is excessive when the throw imparted is as great as 6 in.

The advantages and convenience of short-arm suspension are provided by hanger supports as shown in Fig. 264. This method of suspension, when well constructed, is the most satisfactory means of suspending shaking screens. The hangers have cast-iron marine

type ends with steel bars between. For balancing, the screen is divided into two portions, each portion having a separate driving rod from the eccentric. In order to keep the driving rods straight, one drive is outside the hanger and the other inside.

The shortness of the arm enables the rate of oscillation of the screen to approach the theoretical time of oscillation of a pendulum of length equal to that of the arm. The motion is therefore more nearly a natural motion, and some part of the strains resulting from the frequent changes of direction is absorbed. The motion is not, of course, the regular to-and-fro motion of a pendulum, for the hangers are frequently set at an angle to the vertical and the backward swing is arrested by the eccentric. Similarly, when the hangers at rest are vertical, the motion imparted by the eccentric does not allow the backward swing to develop.

The advantages of short arms, working in bearings, are smooth running and absence of vibration, a saving in power consumption, and a greater ease of alignment.

The driving mechanism and the correct balancing and alignment of screens are important considerations in maintaining an absence of vibration in the buildings. For this reason differential driving motions (as on the Marcus screen) are less satisfactory than plain eccentric or crank drives. The balancing is a difficult matter ; it is best accomplished by driving screens in pairs, with eccentrics on one shaft, but set at 180 degrees ; but the balance is disturbed each time a tub of coal is supplied to the upper screen. Balancing is still more difficult when double- or triple-deck screens are employed for making three or four sizes simultaneously.

Although jigging screens may be considered to be the best type for general use in colliery practice, especially for sizes down to $\frac{3}{16}$ in., on account of their efficiency, high capacity and cheapness, the advantage they hold over other types of screen is appreciably reduced unless they are supplied by a reliable and experienced firm. In such circumstances, they operate without any attention, other than oiling, for long periods. The screen plates will wear without renewal for periods up to two years, and the adjustment of bearing parts is seldom required.

Jigging screens of inferior quality, however, are responsible for many troubles. The moving parts rapidly fall out of alignment, vibration of the foundations is set up, and the building and neighbouring plant is adversely affected. Furthermore the material sometimes tends to bank at one side of the screen and the uneven spreading over the surface reduces the efficiency. Other features of screens not designed with proper care are bad balancing and breakage of the coal, owing perhaps to a too sudden arrest on reaching the end of the shoot on to the screen, to a long fall from the end of the screen, or to a jerkiness of the jigging motion causing fragments to be chipped off by the edges of the apertures.

A means of overcoming the vibrations set up in the building by

jigging screens has recently been described by Jacquelin (*Rev.. de*
The device, described as a " pendular oscillating device of the internal-reacting type," is a system of shock absorbers and consists of two sets of springs attached to the framework of the screen, one set being in tension, whilst the other is in compression.

The capacity of a jigging screen in tons per square foot of screening surface depends essentially upon the sizes into which the coal is to be separated, the proportions of oversize and undersize, the wetness of the coal, the regularity of the feed, the inclination, and the speed and stroke of the eccentric or other actuating device. It is usually considered that the width of a screen fixes its capacity and that its length determines the efficiency of screening. The best guide to capacity is previous experience, but it is usually found that for a capacity of 60 to 70 tons per hour at least 6 ft. of width will be required to remove 1 in. coal or of about 4 ft. to remove 2 in. coal.

Wardell (gives the following figures for the dimensions and arrangement of the holes in perforated plates for screens :—

Diameter of Hole. (In.)	Thickness of Plate. (In.)	Distance between Centres of Holes. (In.)	Weight of Plate. Lb. per sq. ft.
$\frac{1}{4}$	$\frac{1}{8}$	$\frac{3}{8}$	$3\frac{1}{8}$
$\frac{3}{8}$	$\frac{1}{8}$	$\frac{9}{16}$	$3\frac{1}{8}$
$\frac{1}{2}$	$\frac{5}{32}$	$\frac{3}{4}$	$3\frac{7}{8}$
$\frac{3}{4}$	$\frac{3}{16}$	$1\frac{1}{8}$	$4\frac{5}{8}$
$\frac{7}{8}$	$\frac{3}{16}$	$1\frac{5}{16}$	$4\frac{5}{8}$
1	$\frac{1}{4}$	$1\frac{1}{2}$	$6\frac{1}{4}$
$1\frac{1}{4}$	$\frac{1}{4}$	$1\frac{7}{8}$	$6\frac{1}{4}$
$1\frac{1}{2}$	$\frac{1}{4}$	$2\frac{1}{4}$	$6\frac{1}{4}$
$1\frac{3}{4}$	$\frac{1}{4}$	$2\frac{5}{8}$	$6\frac{1}{4}$
2	$\frac{1}{4}$	3	$6\frac{1}{4}$
$2\frac{1}{2}$	$\frac{5}{16}$	$3\frac{3}{4}$	$7\frac{5}{8}$
3	$\frac{5}{16}$	$4\frac{1}{2}$	$7\frac{5}{8}$
$3\frac{1}{2}$	$\frac{3}{8}$	$5\frac{1}{4}$	$9\frac{1}{8}$
4	$\frac{3}{8}$	6	$9\frac{1}{8}$

Vibrating Screens.—Vibrating screens differ from shaking screens in that the reciprocating movement imparted to them is of greater frequency and much smaller amplitude. The oscillation of the surface of a shaking screen is calculated to cause the particles to move forward as a result of the impulses imparted to them. On a vibrating screen, however, the particles usually move forward by gravity alone, and the motion of the surface tends to shake them and

make them dance on the surface in a manner similar to that induced in hand sieving, rather than to cause them to move down the screen.

There are two types of vibrating screen, the impact type and

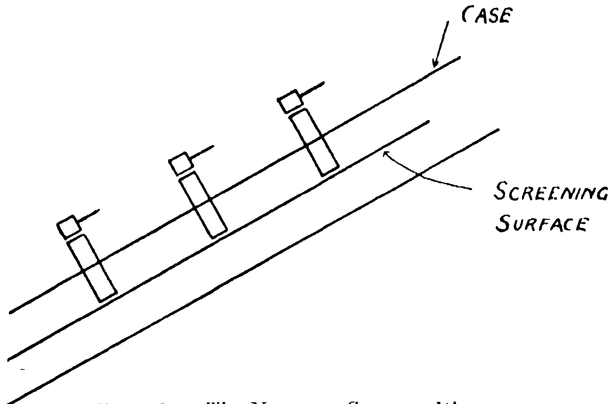


FIG. 265.—The Newaygo Screen : Diagram.

the oscillatory type. In the first class are included the Newaygo, Impact, Leahy, Hoyle, Buzza, and Hummer screens, in which the

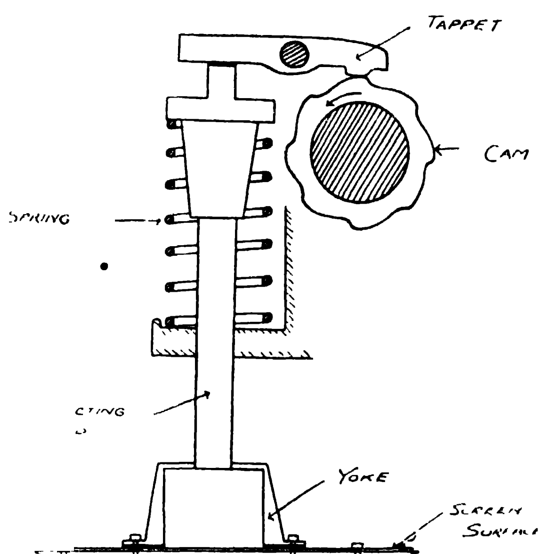


FIG. 266.—Mechanism of Leahy Screen : Diagram.

vibration is induced or terminated suddenly and intermittently, and only the woven-wire mesh of the screen surface is made to vibrate. The second class includes the Arms, Overstrom, Moto-Vibro, and Karlik screens, in which the framework holding the screening surface vibrates rapidly and regularly.

The Newaygo screen consists of a woven-wire surface enclosed in a steel casing, which is pierced by a number of pins. A series of hammers strike rapid blows on the pins, which vibrate the protected screening surface. The principle is shown diagrammatically in Fig. 265.

In the Leahy screen (Fig. 266), made by the Deister Concentrator Co., a revolving cam strikes a series of blows on a pivoted tappet, causing a vibratory motion in a connecting rod. The vibrations are communicated to the screening surface by a yoke fastened rigidly to a cross-piece on the opposite side of the screen surface. In the Hoyle vibrating screen (Fig. 267) oscillation of the screening wires is produced by an eccentric shaft connected by two arms to the screen-

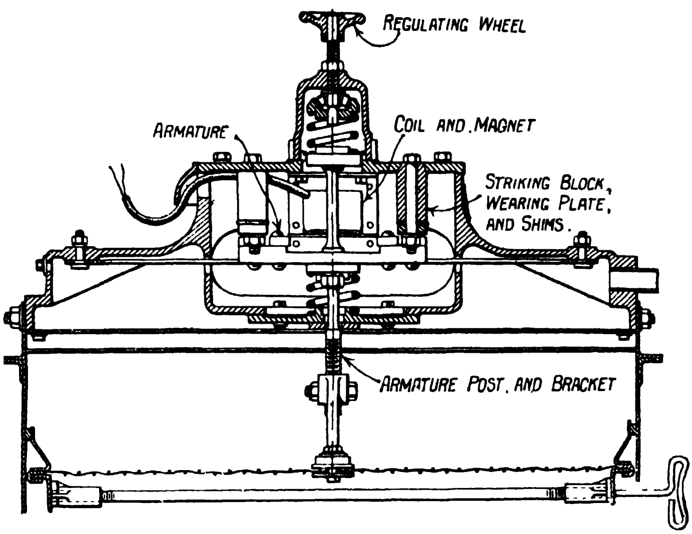


FIG. 269.—The Hummer Screen: Diagram of Vibrating Mechanism.

ing surface. The arms are fastened to contact bars clamped longitudinally along the screen wire. In the Leahy screen, the cross-piece in contact with the wires is placed latitudinally across the surface.

The Hummer screen (Figs. 268 and 269) includes an electro-magnetic arrangement for imparting the vibrations. The screening surface is fastened to a vibrating unit consisting of a solenoid magnet and armature. Current is generated by a special generator supplying 15-cycle, single-phase current, and the magnet alternately attracts and releases the armature, moving it upwards and downwards. On the upstroke, the motion is terminated suddenly by striking against bumping blocks. The amplitude of vibration is regulated by adjusting the tension of a spiral spring to which the core of the magnet is attached.

In all these three screens of the impact type, the wires of the

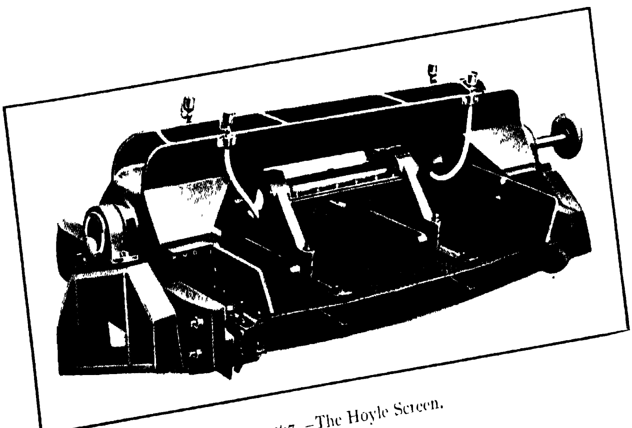


FIG. 267.—The Hoyle Screen.

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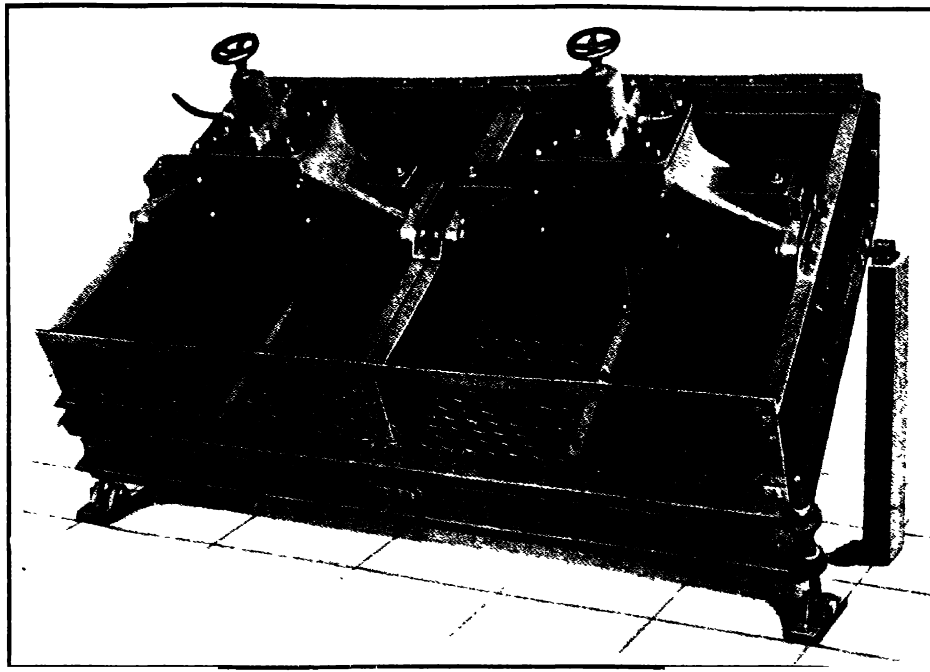

FIG 268 —View of Hummer Screen

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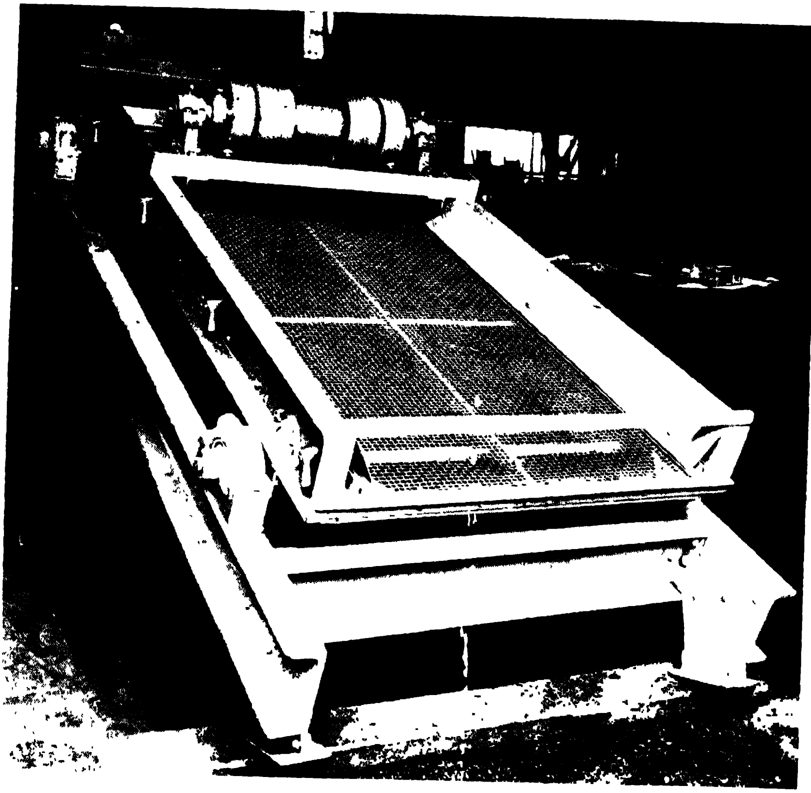

FIG. 270.—View of Overstrom Screen.

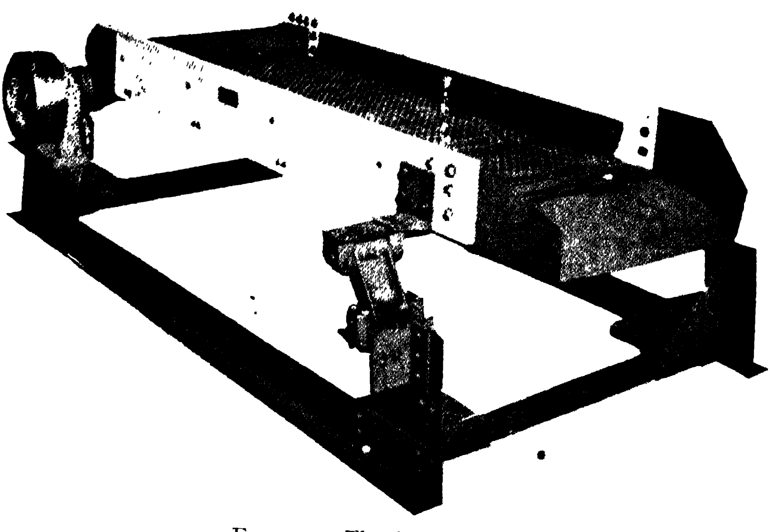

FIG. 271.—The Arms Screen.

[To face page 561.

screen are tightly stretched and vibrate like piano wires, the enclosing frame being relatively stationary. The frequenty of vibration is about 250 per minute in the Newaygo, 1,600 per minute in the Leahy, and 1,800 per minute in the Hummer. All three screens are arranged with a steep slope.

The second class of vibrating screen includes those in which the whole screen and framework moves rapidly to and fro. The Overstrom and Arms screens may be taken as typical.

The Overstrom screen (Fig. 270) resembles the H.H. concentrating table in that it is actuated by the revolution of an unbalanced pulley, running on a shaft fixed rigidly to the frame. The pulley rotates at about 1,600 r.p.m., and imparts an equivalent number of vibrations to the screen. The coal travels over the screen by gravity and is shaken about on the surface by the vibratory motion. The vibrations are so rapid that they are transmitted to the wires forming the screen surface.

The Arms screen (Fig. 271) differs from other vibrating screens in being nearly horizontal, consequently it has the advantage of requiring little headroom. It should, perhaps, be classed as a shaking screen rather than as a vibrating screen, for the oscillatory movement is responsible for the travel of the material over the screen, as well as for imparting a vibration to the screen wires. Nevertheless, its high efficiency must be attributed to the vibrations set up, and, in common with other types of vibrating screens, it is particularly adaptable to screening small coal. It consists of a woven-wire mesh fastened in a steel or wooden framework, which is vibrated by ball-bearing eccentrics, rotating at about 500 r.p.m. At one end the screen is supported by inclined rocker arms, which cause the motion during each stroke to be slightly upward. Against the advantage of small headroom must be offset a slightly higher power requirement as compared with other types of vibratory screen.

Vibrating screens have not been employed at collieries in this country to an extent sufficient to enable a reliable comparison between different types to be made, especially with respect to capacity and efficiency. It may be stated, however, that, as a whole, they have a greater capacity for a given screening area, and work with a greater efficiency than other kinds of screens. This is especially so when they are used on small coal. The rapid vibration not only keeps the screen apertures clear and hastens screening, but also produces a certain amount of classification, there being a tendency for the smaller particles to segregate beneath the larger particles. In these circumstances the small particles are in a position to fall through the apertures more readily. The rapid motion continually brings fresh particles into contact with the screening surface, and the vibration of the individual wires helps the particles to work their way through the apertures.

The capacity of the screens will probably depend more upon the nature of the material fed than upon the nature of the screen,

and any comparison between the two types should be based upon tests on identical material. In general, the capacity will depend upon the relative proportions of coal and shale, of oversize and undersize, and on the accuracy of sizing required. For maximum capacity it is essential that the particles be fed uniformly and at the optimum rate. The load on the screen must not be too heavy to damp down the vibrations (especially with an impact type of screen), nor must it be too light, or the particles will tend to bounce over the surface.

Since the maintenance of a correct depth (or weight) of bed materially affects both the capacity and efficiency of screening, it is obvious that, if the optimum conditions obtain at the feed end of the screen, they will not also obtain at the other end, for much of the material fed will then have passed through the apertures. Screens like the Overstrom and Arms, which utilise a uniform vibration over the whole surface, are unable to overcome this effect, whereas differential vibrations can be applied at different positions on screens of the impact type.

The disadvantage of screens using percussive hammer blows is that the impacts are harmful to the machinery and set up vibrations in the building. An objection to the Hummer screen is that a special generator has to be installed to supply the current for the electro-magnet.

One of the principal uses of vibrating screens is in connection with the dewatering of coal. For this purpose wedge-wire rods are arranged longitudinally along the screen, which is actuated by an eccentric rotating at about 250 r.p.m The screen is nearly horizontal, being inclined slightly upwards from the feed to the discharge end. A modified screen of this type is fitted to all Rheolaveur slurry washing plants, and, in general, the moisture content of the cleaned slurry is reduced to below 30 per cent. There appears to be no reason why some of the other types of vibrating screen should not be used for dewatering, the screening surfaces being made of bronze or brass wire.

For the larger sizes of coal, vibrating screens, with the exception of the Arms screen, are at a disadvantage when compared with shaking screens because of the greater headroom required. They are also more expensive in upkeep because large pieces of coal wear the screen wire much more rapidly than they wear punched steel plate. On small coal, however, there is no comparison between the two. Vibrating screens, though more expensive in initial and upkeep cost, are much more efficient in operation on dry material. It is not established, however, that they are more satisfactory than ordinary shaking screens for screening small wet coal after washing when the bed of coal on the screen can be sprayed liberally with water, and the chief function of vibrating screens in coal preparation would appear to be in connection with the removal of dry dust from coal before washing.

The Berrisford screen, installed at Berryhill Colliery, Stoke-on-

Trent, is an inclined shaking screen, which includes a device for making the screening plates vibrate. The plates are stepped and a screen 10 ft. long consists of four plates. Each plate is turned down for 1½ in. at the lower end and turned up for 1½ in. at the upper end. The turn-up at the upper end of the second plate fits behind the turn-down at the lower end of the first plate, and a space of about 1 in. is left between the lips. When the screen is shaken by the eccentric, the plates are free to shake longitudinally inside the frame by the amounts of the spaces between the lips, and vibrations are set up, which largely prevent "blinding," and greatly increase the efficiency of screening. The screen, therefore, combines some of the advantages of both shaking and vibrating screens, and for dry screening, it is said to have a high efficiency when working with small coal. For $\frac{3}{16}$ in. screens a stroke of 1½ in. is imparted 150 times per minute. A 10 ft. by 5½ ft. screen can deal with 50 tons of coal per hour. The screen is mounted on rollers and driven by an eccentric.

SCREEN SIZE STANDARDS

The absence of any uniformity, either in England or America, in the terminology of coal sizing is due partly to the absence (hitherto) of a demand for accurate terminology, and partly to the difficulty of conforming any agreed system with existing industrial conditions.

The standards of size proposed for the purposes of ore-dressing are inappropriate for coal because, in ore-dressing, the usual unit of measure is the millimetre, and because the sizes are nearly all below 1 mm. In the preparation of coal for sale, however, except on the Continent, the inch is the standard unit of measure and the most important sizes for the purpose of grading the coal are those between ⅛ in. and 4 in. For ore-dressing, Rittinger (*Aufbereitungs kunde*, 1867) proposed a standard scale. He took 1 mm. square as the basic size and increased or decreased the side of the square by $\sqrt{2}$ for successive sizes. The area of aperture in square millimetres was then a power of 2 or of ½. Richards has suggested the use of a $\sqrt[4]{2}$ ratio, which enables a larger number of sizes to be used. The U.S. Bureau of Standards combined these two proposals in its suggested scale for ore sizing, and from the basic size of sieve opening of 1 mm. square, the sizes of the other members of the series were obtained by multiplying by $\sqrt{2}$ until 8 mm. is reached or dividing by $\sqrt[4]{2}$ for the sizes below 1 mm.

The Tyler standard largely used in America and occasionally in England, employs the $\sqrt{2}$ ratio proposed by Rittinger, and the complete series of sizes ranges from 1·05 in. (26·67 mm.) to 0·0029 in. (0·074 mm.).

The standard series proposed by the Institution of Mining and Metallurgy, London, provides that the wire used for making sieves

shall be equal in diameter to the width of the apertures, and that the unit of measurement shall be the inch. Thus, for square apertures whose sides are each $\frac{1}{10}$ in. long, wire $\frac{1}{10}$ in. in diameter is employed. In these circumstances, for each inch of length of the sieve (measured along a wire) there are five apertures, and the sieve is a 5-mesh sieve. With the I.M.M. series, therefore, the size in inches of the particles passing through is half the reciprocal of the mesh number. This is convenient for reference purposes ; in other standard scales there is no such convenience, and the size of the particles passing through the sieve bears no simple relationship to the mesh number.

The sizes of the apertures and the diameters of the wires used in the I.M.M. and Tyler scales are given in Table 147.

TABLE 147.—COMPARISON OF SCREEN SIZES

I.M.M. Standard.				Tyler Standard.			
Mesh No.	Diam. Wire.	Aperture.		Mesh No.	Diam. Wire.	Aperture.	
	In.	In.	Mm.		In.	In.	Mm.
5	$\frac{1}{10}$	0·1000	2·540	3	0·070	0·263	6·680
8	$\frac{1}{16}$	0·0615	1·574	4	0·065	0·185	4·699
10	$\frac{1}{20}$	0·0500	1·270	6	0·036	0·131	3·327
12	$\frac{1}{24}$	0·0467	1·065	8	0·032	0·093	2·362
16	$\frac{1}{32}$	0·0312	0·792	10	0·035	0·065	1·651
20	$\frac{1}{40}$	0·0250	0·635	14	0·025	0·046	1·168
30	$\frac{1}{60}$	0·0177	0·421	20	0·0172	0·033	0·833
40	$\frac{1}{80}$	0·0125	0·317	28	0·0125	0·023	0·589
50	$\frac{1}{100}$	0·0100	0·254	35	0·0270	0·016	0·417
60	$\frac{1}{120}$	0·0083	0·211	48	0·0092	0·012	0·295
70	$\frac{1}{140}$	0·0071	0·180	65	0·0072	0·008	0·218
80	$\frac{1}{160}$	0·0062	0·157	100	0·0042	0·006	0·147
90	$\frac{1}{180}$	0·0055	0·139	150	0·0026	0·004	0·104
100	$\frac{1}{200}$	0·0050	0·127	200	0·0021	0·003	0·074
120	$\frac{1}{240}$	0·0047	0·107				
150	$\frac{1}{300}$	0·0033	0·084				
200	$\frac{1}{400}$	0·0025	0·063				

Holbrook and Frazer (*loc. cit.*) have proposed a scale for coal screening with round, instead of square, holes, varying in diameter from 8 to $\frac{1}{64}$ in. Each successive member of the series has an aperture with a diameter one-half that of the member above it. This series is not altogether satisfactory. A series below 1 in., whose only members are $\frac{1}{2}$ in., $\frac{1}{4}$ in., $\frac{1}{8}$ in., $\frac{1}{16}$ in., $\frac{1}{32}$ in., and $\frac{1}{64}$ in., is inadequate for practical purposes. It provides for four sizes below $\frac{1}{8}$ in., and for

only three between 1 in. and $\frac{1}{8}$ in., which is the reverse of industrial requirements. Moreover, it is difficult to ensure uniformity in the manufacture of punched plates with round holes of $\frac{1}{64}$ in. diameter, and no provision can be made for sizes below $\frac{1}{64}$ in. in a series using round holes.

The object of adopting a ratio of $\sqrt[4]{2}$, $\sqrt{2}$, or 2 to 1 in the standard series is to have the areas of the apertures in the form of a geometrical progression. The only advantage of a geometrical progression of the sizes of screen apertures is that when logarithms of the sizes are plotted as abscissæ, the successive screens are placed at regular intervals. But this arrangement appears to have little practical merit. There is some justification for a regular series of sizes in such industries as, say, the cement industry, and in certain ore-dressing processes, where very fine regular particles are encountered, but when dealing with a material such as pit coal, which contains particles of all and assorted shapes, the advantage would appear to be somewhat questionable.

For industrial purposes, the most satisfactory method of grading coal would be one in which the gradation depended upon differences in the specific surface of the particles (volume divided by surface area) but, unfortunately, this cannot be achieved in practice, especially with such a material as pit coal, in which the shapes are very irregular. Screening is the best alternative, but by screening it is only possible to ensure a fairly uniform gradation, the gradation depending on the maximum or minimum area of the particles in section. It would therefore seem unreasonable to insist on a standard scale of sizes bearing definite mathematical relationships to one another, and some arbitrary series of sizes could be chosen, based on the requirements of industry.

It might be satisfactory to choose a series such as the following :—

4 in.	$1\frac{1}{2}$ in.	$\frac{5}{8}$ in.	$\frac{5}{16}$ in.	$\frac{1}{32}$ in.
3 ,,	$1\frac{1}{4}$,,	$\frac{9}{16}$,,	$\frac{1}{4}$,,	$\frac{1}{64}$,,
$2\frac{1}{2}$,,	1 ,,	$\frac{1}{2}$,,	$\frac{3}{16}$,,	
2 ,,	$\frac{7}{8}$,,	$\frac{7}{16}$,,	$\frac{1}{8}$,,	50 mesh I.M.M.
$1\frac{3}{4}$,,	$\frac{3}{4}$,,	$\frac{3}{8}$,,	$\frac{1}{16}$,,	100 ,, ,,

The sizes of the members of this series bear no uniform relation to each other ; in some parts of the scale the progression is arithmetic, but for the lower members from $\frac{1}{8}$ to $\frac{1}{64}$ in. it is geometric. Whether twenty-four members of the series are necessary is doubtful, but, in industrial practice, there is no need to employ any sizes which have been omitted, and it might be possible to omit several of those suggested.

These sizes would be satisfactory for all coal except export coal, for which gradation in millimetres is required. Sizing in milli-

metres might eventually be acceptable for inland trade, but if the foreign customer requires sizing in his own units, the inland consumer is likely to require sizing in British units. The sizes proposed in South Wales, 80, 55, 25, 15, 8, 4 mm., correspond approximately to 3, $2\frac{1}{8}$, 1, $\frac{9}{16}$, $\frac{5}{16}$, $\frac{5}{32}$ in.

The choice of round or square holes is one made only with difficulty. Round holes ensure greater uniformity of the product, are less liable to " blind," and are cheaper and easier to make. On the other hand, for the smallest sizes, and if wire mesh is used, it is necessary to have square or rectangular holes.

CHAPTER XXX

WASHERY CONTROL

In Great Britain, the control of a coal washery is often entrusted to unskilled labour with only occasional supervision by the colliery engineer, the coke-oven manager, the colliery or coke-oven chemist. Day-to-day testing by the float and sink analysis of samples is usually practised, but the method of collecting the samples is frequently open to criticism and the methods of testing are not always well chosen. In many instances it would appear to be profitable to establish more adequate supervision and to revise the methods of testing.

The general interpretation of float and sink tests has been considered at some length in Chapter II, and the methods of estimating the efficiency of operation of the plant are given in Chapter XXXI. In this chapter we propose to deal more particularly with the application of float and sink tests to washery control.

Washery control may be considered from two points of view, firstly, that of the washery man, who wishes to know whether the washer is operating properly or whether any variation in the quality of the products is occurring, and, secondly, that of the colliery management. The washery man is concerned with the adjustment of the settings of the washer; the colliery management desire to know what proportion of the raw coal is being recovered and whether the washer is working as efficiently as possible.

In this chapter, the question of washery control is regarded from both these aspects; in the next chapter, however, the particular question of efficiency formulæ is considered in greater detail. Here we are concerned with the correct control of the washery, the collection of samples, and the methods of obtaining results relative to the efficiency of operation. The interpretation of those results by the colliery management requires separate consideration.

Control by the Washery Man.—In many washeries, the washer-man judges the efficiency of his washer by the visual examination of snap samples collected from the wash-box or by observation of the washed coal on a conveyor or shoot. The coal is usually black and glistening, and dirt particles are grey or have only a dull black colour. Such methods are, however, of limited usefulness and are only really serviceable to indicate that the washer is working very badly.

The reasons for the inaccuracy of this method are several. When the washery man relies upon visual examination he may easily be misled by the occurrence in the washed coal of dull particles which may actually consist of good coal in the form of durain, or of rela-

tively low-ash middlings, and these he may mistake for shale. He may also observe black glistening particles in the refuse, which he will class as coal, when actually they may enclose a large proportion of shale or pyrites and be, in reality, particles whose rejection is desirable. On other occasions, the shale associated with the coal may be black and, when wet, almost indistinguishable from coal.

The greatest error in drawing conclusions from snap samples collected in a perforated scoop is caused, however, by the fact that much of the dirt retained by the washed coal would not be present in the sample, and that, if present, it would scarcely be noticeable. The dirt retained by washed coal is almost invariably of a very small size, and, when the sample was collected, the water would wash most of the fine dirt particles through the perforations in the scoop Moreover, the fine dirt (less than $\frac{1}{10}$ in. in size) retained in the scoop sample would be indistinguishable from the fine coal.

That the dirt retained by washed coal is usually of this small size is shown by the results of the following three tests, two on Baum washers and one on a Rheolaveur washer. The results are given in Tables 148 to 150.

TABLE 148.—RESULTS OF WASHING A MONMOUTHSHIRE COAL IN A BAUM WASHER

Size (in.)	Raw Coal.				Washed Coal.				Per cent. of dirt removed.
	Per cent of total	< 1·48. S.G.	> 1·48 S.G.	Sinks per cent of total coal.	Per cent. of total	> 1·48 S.G.	Sinks per cent of washed coal.	Ash per cent. in fraction	
$> \frac{1}{2}$ . . .	12	57·4	42·6	5·1	1	0·0	0·0	10·5	100
$\frac{1}{2}-\frac{1}{4}$ . . .	30	71·6	28·4	8·5	24	1·8	0·4	7·9	95
$\frac{1}{4}-\frac{1}{8}$ . . .	18	79·0	21·0	3·8	23	3·6	0·8	6·5	78
$\frac{1}{8}-\frac{1}{16}$ . .	17	79·0	21·0	3·6	25	3·2	0·8	6·5	78
$\frac{1}{16}-\frac{1}{60}$ . .	15	78·0	22·0	3·3	18	4·8	0·9	7·3	74
$< \frac{1}{60}$ . .	8	76·0	24·0	1·9	9	19·6	1·8	22·6	8
Total .	100	73·8	26·2	26·2	100	4·7	4·7	8·5	82

The percentage of dirt removed is about 97 per cent. for the sizes greater than $\frac{1}{4}$ in., but is only 8 per cent. for the sizes less than $\frac{1}{60}$ in. For all sizes the average is 82 per cent. Of the total dirt left in the washed coal (4·68 per cent.), 37 per cent. is less than $\frac{1}{60}$ in. size, and 56 per cent. is less than $\frac{1}{16}$ in. In another test, 46 per cent. of the dirt left in the washed coal was less than $\frac{1}{60}$ in. size, and 72·5 per cent. was less than $\frac{1}{16}$ in. size. The regular day-to-day throughput of raw coal in this washery was 70 tons per hour.

The results in Table 149 are for a Rheolaveur washery dealing with 30 tons per hour of coal through a $\frac{5}{16}$ in. screen.

The average percentage of dirt removed was 94 per cent., and was as high as 82 per cent. for the size $\frac{1}{45}$ in. to 0. Of the dirt

TABLE 149.—RESULTS OF WASHING A DERBYSHIRE COAL IN A RHEOLAVEUR WASHER

Size (in.)	Raw Coal. Per cent. of total.	Raw Coal. < 1·48 S.G.	Raw Coal. > 1·48 S.G.	Raw Coal. Sinks per cent. of total coal.	Washed Coal. Per cent. of total.	Washed Coal. > 1·48 S.G.	Washed Coal. Sinks per cent. of washed coal.	Per cent. of dirt removed.
$\frac{6}{16}$–$\frac{1}{16}$ . .	50	72·0	28·0	14·0	55	0·7	0·4	97
$\frac{1}{16}$–$\frac{1}{45}$ . .	32	65·9	34·1	10·9	31	1·9	0·6	95
< $\frac{1}{45}$ . .	18	68·9	31·1	5·6	14	7·1	1·0	82
Total .	100	69·5	30·5	30·5	100	2·0	2·0	94

left in the washed smalls, 50 per cent. was less than $\frac{1}{45}$ in. and 85 per cent. was less than $\frac{1}{16}$ in. size.

The results obtained in Table 150 (for which we are indebted to Mr. D. R. Wattleworth) were obtained from samples from a Baum washery over a six-day period.

TABLE 150.—RESULTS OF WASHING A CUMBERLAND COAL IN A BAUM WASHER

	Size (in.)	Total Product. Per cent by weight.	Total Product. Ash per cent.	Coal < 1·35 S.G. Per cent by weight	Coal < 1·35 S.G. Ash per cent.	Middlings 1·35–1·60 S.G. Per cent by weight	Middlings 1·35–1·60 S.G. Ash per cent.	Dirt. > 1·60 S.G. Per cent. by weight	Dirt. > 1·60 S.G. Ash per cent.
Raw coal .	> $\frac{3}{8}$	42·9	—	48·0	4·9	30·0	15·6	22·0	61·2
	$\frac{3}{8}$–$\frac{1}{8}$	28·6	—	51·2	3·4	17·8	16·5	31·0	—
	< $\frac{1}{8}$	28·5	—	31·0	3·9	18·0	17·3	51·0	—
	Total .	100·0	25·7	44·1	4·1	23·1	16·1	32·8	—
Washed smalls	> $\frac{1}{4}$	30·0	6·2	77·1	3·5	21·7	16·6	1·2	36·0
	$\frac{1}{4}$–$\frac{1}{10}$	54·4	7·2	75·5	4·2	21·6	17·2	2·9	40·2
	$\frac{1}{10}$–$\frac{1}{30}$	12·4	10·1	71·0	3·9	20·5	17·3	8·5	48·2
	< $\frac{1}{30}$	3·2	22·9	34·0	4·2	34·0	11·8	32·0	56·5
	Total .	100·0	7·7	74·1	4·0	21·8	16·8	4·1	44·7
Washed nuts	> $\frac{3}{4}$	57·9	6·6	82·0	4·2	17·5	16·4	0·5	40·0
	$\frac{3}{4}$–$\frac{1}{4}$	40·4	7·1	79·4	2·5	18·3	17·3	2·3	42·2
	< $\frac{1}{4}$	1·7	8·0	76·8	3·2	19·3	17·9	3·9	52·5
	Total .	100·0	6·7	81·0	3·6	17·8	16·8	1·2	45·1
Slurry .	—	100	24·3	50·8	3·2	21·6	17·2	27·6	68·0
Refuse	No. 1 .	—	—	2·1	—	2·6	—	95·3	—
	No. 2 .	—	—	0·8	—	6·9	—	92·3	—
	No. 3 .	—	—	2·1	—	10·2	—	87·7	—

The dirt in the washed coal was mostly in the smallest size, although the slurry was not admixed, and increasing quantities of dirt were found in decreasing sizes of the washed nuts. Most of the middlings was collected with the washed coal, but some of it passed into the refuse, particularly in the fines rewash box. The slurry contained large quantities of dirt (27·6 per cent. sinking at S.G. 1·60), but the smallest size of the raw coal, through $\frac{1}{8}$ in., was extremely dirty (containing 51 per cent. of sinks at S.G. 1·60).

Expressing the results as percentages of the total sample, the sinks in the washed smalls, comprising 4·1 per cent. of the sample, contain 2 per cent. larger than $\frac{1}{10}$ in., 1·1 per cent. from $\frac{1}{10}$ to $\frac{1}{30}$ in., and 1 per cent. less than $\frac{1}{30}$ in. size.

The figures recorded in these tables indicate the smallness of the dirt particles usually included with the washed coal. These, as already stated, might be lost from a sample in a perforated scoop, and would, in any event, be difficult to perceive.

To overcome these difficulties it would be preferable that the sample should be more carefully selected, and in many washeries it would be worth while for some responsible official to investigate the washery man's method of sampling. It is often left to the man's own discretion, or to that of the yard foreman, but it is a matter of some importance, and should be regarded as such. A responsible official should be able to decide the most accurate method and that most convenient for any washery, and should see that the method, once decided, is adhered to by the washery man.

As an alternative to visual examination, it might be possible to supply the washery man with means of doing simple float and sink tests. A fairly narrow cylinder could be used, and the sample could be shaken up in a liquid of suitable specific gravity. The amount of dirt in the washed coal, or of coal in the refuse, could then be gauged by reading the *volumes* of float and sink. The cheapest solution would be calcium chloride of S.G. 1·4, and sufficient supplies could easily be available. If a litre were thrown away every two tests the cost would be under 1*d*. per test, too small to be considered in relation to the benefit to be derived.

Daily Control Tests.—The washery man's samples must, of necessity, be snap samples, but for the laboratory sample a snap sample is quite unsatisfactory. An indication of the average efficiency of washing can only be based on the careful examination of a representative sample of the whole of the washed coal, or refuse, taken continuously over a long period, or, alternatively, of a series of snap samples of washed coal and refuse, taken at regular and short intervals over a long period.

There are certain times when snap samples would show much lower working efficiency than at other times. For example, in a jig washer in which the excess dirt bed is run off intermittently,

some coal may be discharged with the refuse at the end of the discharge period. If, moreover, the bed is then "light," the pulsations may force dirt particles over with the coal. For these reasons a continuously-discharged bed gives more uniform results than a bed which is run off at intervals, and snap samples are more reliable. A sudden increase in the rate of feeding may cause the bed to build up solidly at the feed end of the box, so that, in effect, a reduced length of the box is used for the actual separation of coal from dirt, although a greater quantity of material is dealt with. Imperfect removal of the dirt may result, particularly if the dirt bed is not run off at a greater rate than the normal, and snap samples would give an unfair reflex of the average efficiency. Similarly, after a stoppage, the bed tends to set compactly, and some time is necessary before it is loosened and normal working is resumed. In a Rheolaveur washer a temporary blockage of a discharge orifice may cause dirt to be removed with the washed coal, and, in all types of washer, abnormal conditions arise.

It will be obvious that snap samples taken during abnormal periods will give misleading results. It is not always easy to recognise abnormal conditions, and it is therefore desirable that samples of the washery products should be taken continuously over the whole day, or that several snap samples should be taken at intervals and mixed to give a composite sample representing the day's work. No such sample can be considered as reliable unless it is made up of individual samples collected, say, each hour. If only a few samples are collected, they should be at a time when the washer is considered to have been working normally for a suitable period. The separate samples should also be of the same weight.

There is another aspect of washery control which should be considered. Hitherto, consideration has been given to the efficient working of the washer, but much may also depend on the efficiency of the washer-man himself. In a Baum jig or a four-trough Rheolaveur washer, a factor of safety is introduced against abnormal conditions, which also reduces the dependence on the human element. With a jig washer in which the refuse is intermittently discharged, much may depend on the attention given to the release of the refuse, and, during the night-time in particular, this may be carried out less thoroughly than at other times. If the washer-man is left to take his own samples, no indication of neglect would probably be given. A very strong case can be stated for the need of some form of automatic sampler which will take samples of the washed coal and the refuse continuously or at frequent intervals. The details of such a scheme will be considered later. At present it is sufficient to emphasise the need of care in taking a representative sample to indicate the average daily working, and that such a sample should not merely represent the washing during five minutes of the working day.

Suitable samples are usually examined by float and sink tests to

determine the amount of coal rejected with the refuse and the amount of dirt included in the washed coal. It has frequently been emphasised that it is not a sound procedure to use, as is sometimes done, a liquid of S.G. 1·3 to determine the loss of coal in the refuse, and another liquid of S.G. 1·5 to determine the amount of dirt in the washed coal, because, in these circumstances, no account whatever is taken of the course of the middlings between 1·3 and 1·5 S.G. The check only indicates the loss of coal of the very best quality in the refuse, and the presence of valuable middlings in the refuse is neglected. It is more justifiable to test the refuse at a specific gravity of 1·5 and the washed coal at a specific gravity of 1·3. This procedure, which is sometimes adopted, debits the washer with the loss of valuable middlings in the refuse, and with the presence of heavy middlings in the coal.

In other cases, a specific gravity of 1·5 is chosen for testing both the refuse and the clean coal. When the coal contains practically no middlings, and is therefore easy to wash, this seems to be a reliable procedure, but in our opinion the only cases in which it is really advisable is when the raw coal contains less than 5 per cent. of material of specific gravity between 1·35 and 1·6. Practically all modern washers, installed by reliable firms, can separate the best coal from "pure" shale, and little loss of good coal in the refuse may be anticipated. Without careful control, however, the most efficient recovery of middlings will not be accomplished. If the tests on the washer are performed at only one specific gravity, it is impossible to distinguish between the three constituents of ordinary raw coal—namely, "pure" coal, middlings, and shale. To check the course of the middlings, it is essential to use liquids of at least two different specific gravities.

The choice of the specific gravities of these two liquids is a matter for decision at each individual washery. For the lower specific gravity, 1·35 is convenient and generally suitable, and for the higher specific gravity 1·60 is usually satisfactory.

When the two specific gravities have been chosen, the washer should be made to work as nearly as possible to a specified distribution of coal, middlings, and refuse. Assuming that specific gravities of 1·35 and 1·60 have been adopted for the two testing liquids, the best washing results for a coal containing 15 per cent. of middlings (of S.G. 1·35 to 1·60) might be represented by some such figures as the following :—

	Floats at S.G. 1·35.	Floats at S.G. 1·35 to 1·60.	Sinks at S.G. 1·60.
Washed coal	—	$>$10 per cent.	$<$1 per cent.
Refuse	$<$1 per cent.	$<$ 5 per cent.	—

The actual figures will, of course, vary from coal to coal, and according to market conditions, but as a general rule it will probably be found to be satisfactory to make the washed coal contain more than twice as much of the middlings fraction as the refuse.

In Table 151 the amount of middlings in a number of coals is shown. The middlings are assumed to sink at S.G. 1·35 but to float at 1·60. The ash contents are also recorded.

TABLE 151 —FLOAT AND SINKS TESTS ON UNWASHED COALS.

Coal.	"Pure" Coal, < 1·35 S.G.		Middlings, 1·35–1·60 S.G.		Dirt, > 1·60 S.G.	
	Per cent. by wt.	Ash per cent.	Per cent. by wt.	Ash per cent	Per cent. by wt.	Ash per cent.
Durham i. .	77·7	2·8	13·0	15·8	9·3	68·2
,, ii. .	71·3	1·8	9·7	13·1	19·0	68·4
,, iii. .	75·4	2·1	8·7	18·0	15·9	64·0
,, iv. .	78·1	1·5	7·4	16·0	14·5	67·2
Lancashire i. .	20·2	3·3	23·4	19·6	56·4	52·3
,, ii. .	77·2	1·5	4·0	18·2	18·8	73·7
,, iii. .	90·8	1·5	1·1	17·6	8·1	62·2
,, iv. .	55·3†	3·3	4·4	18·0	40·3	82·1
Yorkshire i.* .	48·2	2·6	33·5	12·4	18·3	60·8
,, ii. .	79·8	1·4	3·5	22·3	16·7	86·8
,, iii. .	70·9	2·0	11·3	14·3	17·8	67·0
,, iv. .	70·6	1·6	8·7	9·7	20·7	70·9
,, v. .	75·9	2·5	7·5	14·8	16·6	69·3
,, vi. .	66·5	4·3	13·9	15·2	19·6	59·9
Cumberland i. .	43·4	4·1	21·9	16·4	34·7	61·2
,, ii. .	69·9	2·5	10·4	14·4	17·7	68·5
Staffordshire .	59·9	1·9	9·6	16·0	30·5	78·3
Australia . .	75·4	5·7	14·2	19·3	10·4	72·0
India . .	26·8	7·0	67·2	17·5	6·0	48·4

* Nut coal; the other coals are of 1 in. or less down to dust.
† Floats S.G. 1·40

Assuming that fair average samples may be taken and that they should be tested in liquids of S.G. 1·35 and 1·60, it is necessary to consider where, and by whom, the tests should be carried out, and the method of procedure. It is desirable that the tests should be made by persons who are either responsible for the efficiency of washing (the washery manager) or by independent workers (colliery or coke-oven chemists), who report to those in charge. It is seldom that a shift hand himself is fitted for the task of testing and, as such tests are sometimes necessary to determine the efficiency of personal control as well as of the washer itself, it is desirable that the tests should be made by more responsible, or by independent, people.

With a washer dealing with 1,000 tons of coal per day for

300 days per year, 300,000 tons of coal would be put through the washery and, with the removal of 20 per cent. of dirt, 60,000 tons of refuse would be rejected. If the coal lost in this refuse could be reduced from, say, 3 to 2 per cent. through the proper application of efficient testing to the regular working of the washery, an additional 600 tons of coal per annum would be recovered, worth, say, £360. It will be seen that the saving on loss of coal in the refuse, due to efficient control, would alone make it worth while to employ a washery manager. Such a man should also be able to improve the regular quality of the washed coal, and to reduce the possibility of stoppages due to breakdowns, by better supervision of plant, feed arrangements, wagon supplies, etc. A washery involves a somewhat different type of control to that necessary on a coke-oven plant, or at a colliery, and the particular duties can only be carried out suitably by a man specially appointed and responsible for the work. In some washeries, a washery manager is more interested in the mechanical working of the washery, although he is also responsible for the working results. Such a type of man may take unkindly to scientific testing, and, in such a case, the manager may supervise the taking of samples and have the tests made by chemists at the colliery or coke-oven laboratories, or have a routine chemist of his own to make the tests for him. In most cases, however, a responsible washery manager would soon learn to carry out the simple float and sink tests which are necessary.

Another arrangement is to transfer a man who has been trained in the laboratory to the washery when he has attained a suitable age and sufficient ability to be responsible for its efficient working. The mechanical repairs would be left in other hands. Such a man can then make the tests, and can see to the making of any adjustments himself.

When the tests are made in the washery building—or even in the laboratory—they should be simplified as much as possible. It may be desirable to have the samples tested quickly, and, in this event, the wet samples cannot be tested in aqueous solutions. In any event, calcium chloride cannot be used for a density higher than about 1·40, and zinc chloride (which can be made into solutions for specific gravities up to 1·75) is much more expensive and has no advantage in point of cost over suitable organic liquids. Organic liquids are preferable in being immiscible with water, and wet samples can therefore be used without decreasing the effective specific gravity of the separating liquids. Chloroform (S.G. 1·48) is convenient and quick to use, but the price of carbon tetrachloride (S.G. 1·61) is only one-third of the price of chloroform and, diluted with benzol if necessary, is the most suitable liquid to use. It displaces water from a coal surface by reason of its superior wetting power for coal, and the coal floated from the refuse is, in effect, dried. The dirt in the washed coal is, however, wet, since water wets it more easily than organic liquids, and the water present after washing is not displaced

by the carbon tetrachloride. The amount of dirt present is, however, only small, and it can be spread out and air-dried in a few minutes. It is, of course, always better to air-dry the samples before testing, even with organic liquids, and the expedient of testing wet samples should only be resorted to when the results are required immediately.

The procedure of testing varies according to whether wet or dry material, or large or small particles, are being tested. Particles greater than 1 in. in size can be placed one at a time into a beaker containing the liquid, and particles from 1 in. to about $\frac{1}{8}$ in., may be fed in small handfuls. Each piece, or each handful, should be thoroughly immersed and stirred before the floatings are removed. With particles below $\frac{1}{8}$ in. size, not more than 30 gm. should be treated at one and the same time, to prevent float particles from being trapped in the sinkings and *vice versâ*; after the liquid has been thoroughly stirred, a few minutes should be allowed for settling. Very fine particles may need to stand for an hour or more before they are adequately separated, and care should always be taken that there is a clear band of liquid between the upper layer of floatings and the lower layer of sinkings.

The quantities recommended for use under the best conditions are approximately as follows :—

Size (in.)	Weight.
Under $\frac{1}{8}$	30 gm.
$\frac{1}{8}$–$\frac{1}{4}$	200 ,,
$\frac{1}{4}$–$\frac{1}{2}$	1,000 ,,
$\frac{1}{2}$–1	32 lb.
1–2	2 cwt.
Over 2	4 ,,

With particles greater than 1 in. in size most of the "pure" coal and "pure" dirt particles can be picked out by hand, and only the doubtful ones need actually be tested. This selection of particles should, however, only be done by an experienced man, and it is often misleading, for a particle of apparently pure coal may hide a lump of pyrites inside it. Washery control is usually more important for the small sizes less than $\frac{1}{8}$ in. size, for this material is the most difficult to wash. The larger quantities advisable to obtain a representative sample of material over $\frac{1}{2}$ in. size really offer little difficulty in testing, but, at the same time, such large samples cannot easily be treated daily. It is usually sufficient to make an occasional analysis of the larger sizes, and for unsized material less than $\frac{1}{2}$ in. a sample of at least 500 gm. should be used. If a sample of only 100 gm. is used, the floatings recovered from the refuse, or the sinkings from the washed coal, may only be of the order of 1 gm., and the percentage errors in rough weighing may make the result rather misleading. Another error may be introduced since a 100-gm.

sample is often too small to compensate for the possible errors in reducing the sample to this quantity. The results of six daily tests using 100-gm. samples may give an average weekly result which is correct, but each daily sample is subject to errors, and conclusions cannot safely be drawn from such samples. It is worth while to use the larger sample of 500 gm. and thus to make each daily analysis more reliable.

In carrying out the tests, the washed coal should be placed in the liquid of S.G. 1·35 and the floatings removed. The sinkings should be dried (a few minutes is sufficient if an organic liquid is used as the separating liquid) and placed in the liquid of S.G. 1·60. The floatings and sinkings at this density should be collected, weighed and recorded as " middlings in washed coal " (sink at 1·35, float at 1·60) and " dirt in washed coal " (sink at 1·60). In testing the refuse, the bulk sample should be placed in the heavier liquid first and the floatings transferred to the lighter liquid. The products should be recorded as " middlings in refuse " (float at 1·60, sink at 1·35) and " coal in refuse " (float at 1·35).

When the samples for testing are used without previous air drying, allowances must be made to reduce all weights to a given moisture content. As a rule, assumptions based on a periodical moisture determination may be made to correct for the moisture contents of the washed coal and the refuse. The use of wet samples is not to be recommended, however, unless immediate results are required. Apart from errors introduced by the varying moisture contents, the separating liquids become dirty and more difficult to use. For routine tests the samples can easily be spread out in a suitable place overnight to air-dry, and may be tested on the following morning; alternatively, the samples can quickly be dried in an oven.

Sampling.—Float-and-sink tests, however carefully performed, are an unreliable guide unless reliable samples are used for testing. Sampling is an empirical operation, of which the principles are but rarely appreciated, it is therefore necessary to consider this branch of washery control in some detail.

To appreciate the difficulties of sampling, it is helpful to realise that 1 ton of coal is equivalent to over 1,000,000 gm., and if a test is dependent on the use of, say, 100 gm. selected from an output of 100 tons per hour, the quantity tested represents only one-millionth part of an hour's output. It is necessary that very special precautions should be taken in order to make the final sample used for float-and-sink tests truly representative of the bulk. A sampling " sense " must be developed by the sampler, and this is only possible when he is familiar with certain facts and principles on which standard sampling methods are based, as well as with approved methods of procedure in sampling different brands of material.

It should be remembered that, even in one seam of coal,

variations in composition occur between samples taken from different horizons; bright coal (clarain and vitrain) generally contains less ash than dull, hard coal (durain), or dull friable coal (fusain). Apart from the ash which is fixed in the coal material, the chief contribution to the average ash content of raw coal is made by free dirt from partings, or from the roof, or floor, of a seam. A knowledge of the physical characteristics of the different types of shale from the pit, viz., colour, shape and density, are useful to those interested in washery control.

Effect of Handling.—It is important to remember the effect of handling coal in bulk in trucks, shoots, etc., and in loading and unloading generally. The shape of pieces of coal less than 3½ in. in size is often roughly cubical and pieces of shale are much flatter. Consequently coal particles will tend to roll down the free surface of a shoot, or a heap (in a hopper, for example), when the angle of inclination is greater than that of repose. Shale, on the other hand, will not roll readily and generally slides down a steeply-inclined surface. Even when coal and dirt both slide, the dirt moves more slowly than the coal because of its greater coefficient of friction. In this way (as is well known to be the case on spiral separators) a partial separation of coal and dirt may occur wherever raw coal is handled on an inclined surface. This fact is utilised by pickers round shale tips, for the material most resembling coal (usually intergrown coal and shale or pyrites) is concentrated round the perimeter of the base of the tip.

A partial separation of coal and dirt, and a concentration of material according to size, may also occur if several layers of particles are formed in a shoot. The largest coal particles, with their tendencies to roll, are less restrained by frictional resistance than the large dirt particles which tend to slide. When the direction of travel is changed abruptly through a right angle, the largest and cleanest particles of coal may be found at one side of the stream of material in the second shoot, with the largest particles of dirt at the opposite side.

It is also worth recalling that, in a wagon of unwashed coal, the finer sizes (containing usually the greatest amount of dirt) settle to the bottom of the wagon, and they are helped to do this when shunting operations are frequent, or if the distance travelled by the wagon is great. For this reason samples taken from the top of a wagon of dry unwashed coal are unreliable.

Bulk Ratio.—It is important to know what is the least amount of material which may be taken as a representative sample, or, in other words, what is the "bulk ratio." For run-of-mine coal, 600 to 1,200 lb. are generally recommended to represent a bulk of from 10 to 1,000 tons. The bulk ratio chiefly depends on the size of the dirt. This may be illustrated by a table (due to Illingworth) in which it is assumed that two independent samplers choose samples truly representative of a cargo, and their samples agree with the

exception of one piece of dirt, assumed to be 4 in. cube, and containing 80 per cent. of ash.

TABLE 152.—EFFECT OF SIZE OF SAMPLE ON LIMITING ERRORS OF SAMPLING

Weight of Sample (lb.).	Per cent. Ash in Sample.	
	Shale included.	Shale excluded.
100	9·7	6·0
200	7·8	6·0
400	6·9	6·0
600	6·6	6·0
1,000	6·4	6·0

This is not an example which is directly comparable with washery practice, but it emphasises the dependence of the amount of sample necessary on the size of the dirt. In Table 153 the effect of one piece of shale of different sizes, included or excluded in samples of 100 lb., is recorded. The ash content of the normal sample (when the extra piece of shale is excluded) is 6·0 per cent. The extra lump of dirt is assumed to be cubical and to contain 80 per cent. of ash.

TABLE 153.—EFFECT OF SIZE OF DIRT ON SAMPLING ERRORS

Size of Dirt (in. cu.).	Weight of One Piece of Dirt (lb.).	Variation of Ash per cent.
4	5·00	3·70
3	2·10	1·51
2	0·64	0·46
1	0·08	0·06
¾	0·03	0·02

It will be observed that, with a 100-lb. sample, it is only with dirt of over 1 in. that appreciable errors may be caused by the incorrect inclusion of one piece.

The most thorough work on the bulk ratio for sampling has been done by the United States Bureau of Mines. Two samplers each took a 100 lb. sample of a cargo of 1,000 tons of coal, and later reduced it by the same procedure to 3 lb., from which an analytical sample was taken. Each sampler collected further 100 lb. samples,

each of which was reduced afterwards to 3 lb., and mixed with the first 3-lb. sample. This procedure was followed until a gross sample of 1,200 lb. was obtained. The results of the analyses of the cumulative samples (Fuel, 1927, 6, 394) showed that a sample of from 800 to 1,200 lb. must be taken if duplicate samples should not differ by more than 1 per cent. in ash content.

A sample of unwashed coal through 3½ in., or through 2 in., to represent, say, a day's feed to a washery, should preferably not be less than 1,000 lb.; where the maximum size is 1 in. a sample of 250 lb. is sufficient, and not more than 100 lb. is necessary when the maximum size is ¾ in. This is illustrated in the following table (adapted from Bailey, *Journ.* Ind. Eng. Chem., 1909, 1, 161; 1910, 2, 543).

TABLE 154.—VARIATION OF SIZE OF SAMPLE ACCORDING TO SIZE OF LARGEST PIECE

Weight of Largest Pieces (lb.).	Size of Largest Pieces * (in.).	Smallest Bulk of Sample (lb.).
6·70	5¼	39,000
2·50	3¾	12,500
0·75	2½	3,800
0·38	2	1,900
0·24	1¾	1,200
0·12	1⅖	600
0·04	1	230
0·02	¾	90

* Pieces assumed to be cubical and of S.G. 1·3.

Actually, for coal greater than 2 in. cube, such large samples need not usually be taken, since the large dirt of that size is easily detected, and would be removed at the picking belts. The largest samples which are commonly taken in practice are 1,200 lb., for run-of-mine coal. The chief point of interest in this table is the smallness of the sample necessary for reliable sampling when the largest size is 1 in.

Except in washery control, coal samples are assessed mainly on a basis of ash content. For ash analysis, the bulk sample may be crushed, and the gross size of the sample can be reduced by standard method of coning, quartering, and the rejection of opposite quarters. This can go on progressively. For washery control, however, the coal samples must not be broken up, for the efficiency of washing is judged by the removal of free dirt, and crushing may liberate dirt from middlings. On the other hand, a sample of 1,000 lb. cannot be used for routine float-and-sink tests. Since

most interest attaches to the smallest sizes of coal and dirt (for the large dirt is easily removed), it is permissible to sieve the large sample of raw coal over a ½-in. screen (or the same size of screen as is used to separate the smalls for final washing or rewashing). The oversize may be examined separately, and this is usually conveniently done by picking out the dirt by hand. Doubtful pieces can easily be checked by dropping in a suitable separating liquid. The undersize may be well mixed and reduced in quantity to 125 lb. This reduced sample may then be treated by successively mixing, coning, quartering, and rejecting opposite quarters, until a laboratory sample of about 5 lb. is obtained. To conform to standard methods of preparing a sample for analysis (as when determining the ash content of a consignment of delivered coal) the sample through ½ in. size should not be reduced below 125 lb. without breaking the largest pieces so as to pass through ¼ in. mesh, but since this is not possible for washery testing, it is all the more important to pay unusual care in mixing before each successive reduction in bulk of the gross sample. A quantity of not less than 1,000 gm. should be used for float-and-sink tests on the unwashed coal.*

The procedure outlined is based on the accepted methods for sampling large quantities of coal whose maximum size is over 1 in. In washery practice, it may be necessary to sample the unwashed coal only at rare intervals, and a procedure similar to that described is necessary to obtain a fair sample. When daily samples are taken at a washery which has a regular quality of feed, it is satisfactory to take smaller gross samples and average the results over a suitable period. In this way each daily sample is liable to an error, but the average result will be nearly the same as when one large gross sample is taken. The work of handling one large sample, and making one series of float-and-sink tests, may prove much simpler than dealing with a number of small samples and carrying out a large number of float-and-sink tests.

Slack.—Unwashed slack, through ⅝ in., is an easy material to sample, and, from considerations previously given, it will be obvious that a sample of only 100 lb. need be taken as the gross sample. Care is necessary in reducing the gross bulk of the sample, owing to the possibility of segregation of coal and dirt particles during coning. Washed slack is usually free from the larger sizes of dirt, but since the small quantities which remain are of the utmost importance in washery control, great care is necessary to obtain a representative sample. A gross sample of less than 25 lb. may give misleading results, and 50 to 100 lb. should be collected.

As previously mentioned, it is difficult to obtain an average sample from the wash-box itself, and it is best to take the sample from the shoot discharging the washed coal from the drainage

* The quantity of 500 gm., previously recommended for coal through ½ in. size, referred to washed coal samples which contains fewer dirt particles and is of fairly uniform composition.

elevator into hoppers or on to a scraper conveyor, or as the conveyor discharges the coal into hoppers or wagons. When the coal is taken from the top of a conveyor, segregation may have occurred; moreover, when such a sample is taken by hand, there is a tendency for the hand to retain the coarser particles, leaving the finer particles behind. Although " hand " samples may be reliable when the coal is dropping freely, it is advisable to use a scoop or a shovel when sampling coal from a conveyor, or from any other place where the coal is not falling freely. The gross sample may then be well mixed, coned and quartered, and reduced to the quantity necessary for the laboratory sample.

Washed Nuts.—Nuts, beans, and peas are usually fairly clean and of uniform size after washing, and are easy to sample. A few shovelfuls taken from various parts of the wagon, or, preferably, from the stream as the wagon is being loaded, will usually give a representative sample for a coal which is easily washed. If the raw coal contains a large middlings fraction, a larger sample is necessary, particularly for the biggest nuts.

Refuse.—It is not always easy to obtain a representative sample of refuse from a washery. In the first place, the largest and heaviest particles of dirt settle at the feed end of the wash-box, or trough, and the smallest and lightest particles settle at the delivery end. Separation according to density and size has therefore occurred before the dirt is discharged. The dirt which is small enough to pass through the fixed sieve of the wash-box, may also be segregated, according to size and density, before it reaches the screw conveyor which removes it from the wash-box. Further segregation may occur in elevating, and in discharging the material down shoots.

In a Baum washery it is often convenient to sample the refuse from each dirt elevator. In a washer with a capacity greater than 75 tons per hour, the first of the two wash-boxes has a dirt elevator at each end, and the rewash-box has an elevator at the discharge end. A sample taken from the buckets will only be from the surface of the material in them, and may consist mostly of the smaller sizes, on account of the segregation which may occur. It is also difficult to obtain a truly representative sample from the scraper conveyor on to which the elevators frequently discharge, since the dirt is not well mixed.

When the daily sample is taken from the elevator buckets, the difficulties of obtaining a representative sample should be borne in mind. In any event, it is desirable to make a check test on the total refuse as it is discharged from the scraper conveyor or into the dirt hopper. A gross sample of at least 100 lb. should be accumulated, well mixed, and reduced in bulk, by the standard method.

In a Humboldt washery it is usually possible to obtain a sample of the refuse as it is discharged from a shoot into the refuse hopper. The refuse includes the dirt from both the " rough " and the " rewash " boxes. It is usually possible to decide, by the maximum

size of the particles, from which box the refuse comes, and since the refuse is discharged from the wash-boxes intermittently, samples of the dirt from both boxes can be collected. Each sample should be of about 100 lb. bulk, and should be mixed, coned and quartered on the floor or on a board kept handy for the purpose. If only one sample of refuse is taken, it should be large (say, 200 lb.) and taken over a longer period to compensate for the irregularity of discharge of the dirt bed from each box.

In a Rheolaveur washery there is no difficulty in obtaining a representative sample of refuse from the final rheo-boxes discharging into the refuse trough. Samples of dirt from each discharge orifice may be collected. In Draper and Robinson washeries, and on concentrating and dry-cleaning tables, the refuse is easily sampled

Discretion must, however, be used. In sampling the refuse from a concentrating table it is not sufficient to pass a bucket slowly across the refuse-discharge collector ; the weight of the samples taken at different points must correspond exactly with the proportion of the total product discharged at those points, and a special collector should often be used, taking a sample simultaneously across the whole width of the refuse (or washed coal) discharge area.

In a trough washer the heaviest dirt is deposited near the feed end of the trough and the lightest middlings are deposited near the delivery end. With a simple trough washer the long length of the trough makes it difficult to collect a representative refuse sample. It is therefore advisable to wait until the refuse has been collected in tubs, ready to be thrown on the tip ; the tubs can then be sampled by collecting portions from different levels. Alternatively, the place where the wet dirt has been recently deposited on the tip may be recognised, and a representative sample taken from this material. If this procedure is followed, the owners of trough washers will frequently be surprised to find that as much as 10 per cent. of coal is being rejected with the refuse from a simple trough washer, although dirt samples, taken from a point near the feed end, may show only 1 or 2 per cent. of free coal.

Consideration has been given to the methods of sampling different types of material, and also to the variations necessary with various kinds of washers. It is not out of place to consider how the gross samples should be treated to obtain the samples for testing. It has been pointed out that it is necessary to collect a larger gross sample than is frequently thought to be desirable. These gross samples should be sampled down near the place where the samples are collected. Sampling boards may be arranged in convenient places.

Coning and Quartering.—The sample should be well mixed with a shovel by being turned over several times. It may be convenient to have two boards—or one large one—for this purpose, and the sample may be built up into one cone and then built up into another. Care should be taken to avoid fracturing the particles during coning and quartering, or interstratified particles may be separated into

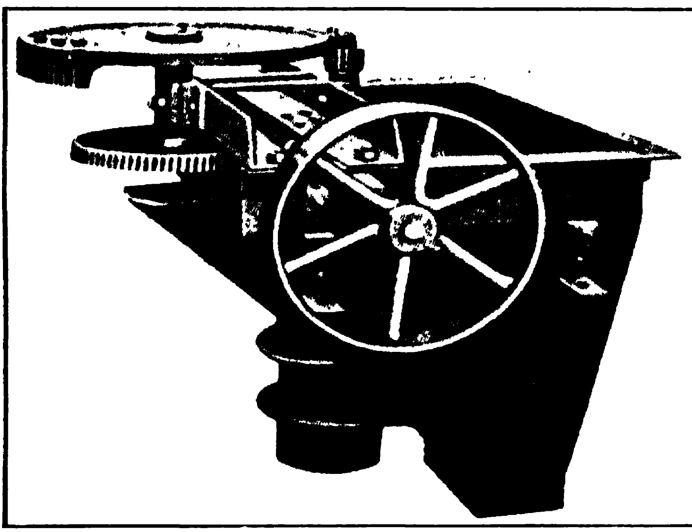

FIG. 272.—Mechanical Sampler.

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coal and shale and thus lead to erroneous deductions being drawn from subsequent float-and-sink tests. Each shovelful is made to fall evenly on the apex of the cone by a suitable jerking motion of the shovel. Much of the coarser material will slide or roll to the base of the cone, and the smaller material will be retained in the centre of the cone, which should be of regular form to preserve the uniformity (the proper proportions of the different sizes) in all sections. The cone is then flattened by pressing the shovel on the apex, and giving it a circular motion, until the mass is spread into an even layer. This is divided into quarters by boards bisecting each other at right angles at the centre of the layer. Opposite quarters are rejected and a new cone is made, and the same procedure followed until a sample of about 5 lb. is obtained. A gross sample of 100 lb. would therefore be coned and quartered four times until two opposite quarters gave a final sample of about 5 lb. It cannot be too strongly emphasised that this rather long procedure is an essential preliminary if washery testing is to have any real significance, and is not merely to be a series of compulsory tests, to be made as quickly as possible, without regard to any rational method of procedure.

Mechanical Sampling.—The adoption of mechanical methods of sampling greatly simplifies the procedure, for not only is the sample representative of the average results over an extended period, but it is taken without personal bias, and requires no attention. No doubt the non-adoption of such methods of sampling has been due to lack of data on their working and on their ability to withstand severe usage.

The automatic mechanical sampler illustrated in Fig. 272, is a modification of the Vezin sampler which is much used in ore-dressing practice. The sampling device is fitted at the top of an elevator or in some other suitable place. A pulley wheel drives a worm shaft which actuates a toothed wheel connected to a disc above it, carrying teeth only in one segment of the disc. Periodically, the teeth on the disc engage with a small cog wheel, which causes the sampling mouth to rotate through the stream to be sampled. In the photograph, the mouth of the sampler is underneath the toothed wheel.

This sampler may be connected to a shaft, and by suitable choice of gearing, snap samples may be collected at long or short intervals. The mechanism is robust, and therefore suitable for coal washeries.

Another mechanical sampling device is illustrated in Fig. 273. It can readily be made without much expense. This device has been fitted in the shoot at the top of a refuse elevator by Mr. R. L. Cawley, of Whitehaven. It may also be fitted at the top of a washed coal elevator. It consists of three lengths of 3-in. piping, which are connected by pieces of ⅜-in. plate, forged to fit the piping. The pipe sections are staggered so that the wall of each lower section is in line with the centre of the higher section. In this way the stream of material passing full bore down the first section, is cut by the top

of the second section, and only a proportion of the stream passes through the second section. A second cut at the top of the third section still further reduces the size of the sample, which finally passes through the third pipe section to a suitable collecting box.

In the original device, the two upper pipe sections are made 12 in. long, and the third section is made of any convenient length. The gap between the first and second, and between the second and third sections is 3 in. The connecting plates are arranged so that they prevent the entry of material from the elevator casing into the

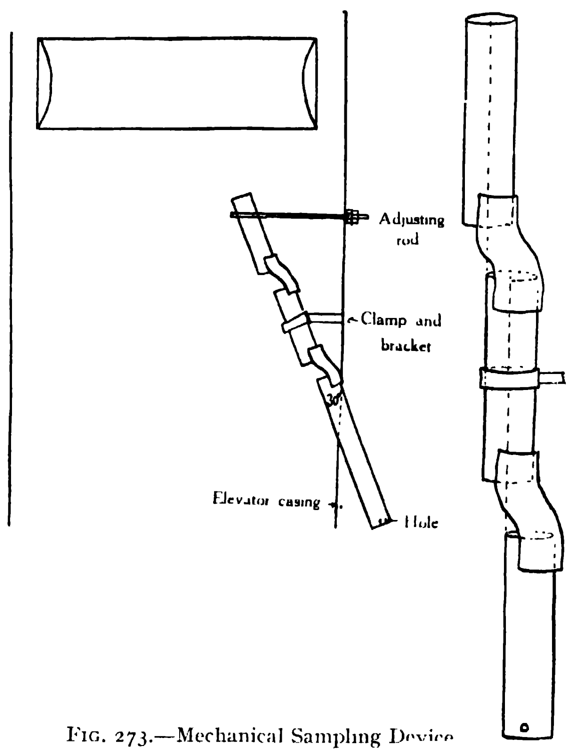

FIG. 273.—Mechanical Sampling Device

second or third sections directly, thus ensuring that only material which has passed through the first section passes to the succeeding sections.

The sampling device is fastened to the elevator casing by means of a bracket and a clamp round the middle section. A second clamp round the top section is fitted to a threaded rod which passes through the elevator casing and is fastened by a nut. The whole sampling device is then sufficiently inclined to the vertical to prevent most of the material entering the upper pipe from dropping through the subsequent sections without being "cut." As the angle of inclination to the vertical is increased, the material leaves the upper pipe in a

less compact mass and facilitates the cutting of the stream. If the sampling pipes are inclined too much to the vertical the pipes tend to choke. A compromise has therefore to be effected to enable the material to flow freely through the pipes but to allow a clean cut to be made at each successive section of piping. A suitable angle is about 30 degrees to the vertical. When a suitable angle has been chosen, the inclination is fixed by adjustment of the nut on the upper threaded rod.

When the material flows freely through the pipes, the size of the sample taken is governed by twisting the lower pipe sections about the centre line of the upper section. For this purpose a rod is inserted in a hole at the bottom of the third pipe and moved in a clockwise or anti-clockwise direction. In this way the cutting edges of the lower pipes are moved about the centre of the upper pipe and more or less material leaves the lower pipe. The material rejected by the sampler is collected in the elevator, and only the sample passes outside. It was found that a sample of about 2 cwt. of refuse was collected in eight hours. This is a suitable quantity to collect, and the laboratory sample prepared from it should enable the results of testing to be accepted with greater confidence.

CHAPTER XXXI

WASHERY CONTROL: EFFICIENCY FORMULÆ

ONE criterion upon which the saleability of a coal is judged by a colliery company is its ash content. Provided that the ash content is below a certain figure, the colliery sales department knows that the coal can usually be sold.

In producing a coal with not more than this maximum ash content, the colliery company wishes to keep its loss of output, in the form of washery or picking dirt, to the minimum amount. A formula to express washery efficiency should be able, satisfactorily, to inform the colliery officials whether this is or is not being done.

The value of a formula for washery efficiency is well expressed by Frazer and Yancey (*Trans.* Amer. Inst. Min. Eng., 1923, 69, 447), who say that such a formula " is most useful in the continuous study of the operation of a given plant, in the routine adjustment of the plant from day to day, in the development and improvement of the plant, and in determining changes in the quality of the raw coal mined."

Various formulæ for judging washery efficiency have been put-forward from time to time,* but those of Drakeley and of Frazer and Yancey only need be considered.

Drakeley's formula is based entirely upon float and sink analyses of the raw coal, the washed coal and the refuse. He obtains two expressions, termed the Qualitative Efficiency (or efficiency of dirt removal) and Quantitative Efficiency (or efficiency of recovery of coal). By multiplying these two figures, a value for general efficiency is obtained. Drakeley suggests the use of a liquid of S.G. 1·35 to determine the amounts of the raw coal, washed coal and refuse floating in it.† Using Ff, Fw and Fr to denote the percentages of each sample floating at S.G. 1·35, his two formulæ may be written as follows :—

$$\text{Qualitative efficiency} = \frac{100\,(Fw - Ff)}{100 - Ff} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (47)$$

$$\text{Quantitative efficiency} = \frac{100\,Ff - Fr \times \text{per cent. of refuse}}{Ff} \quad . \quad (48)$$

* Hancock, " Ironmaking in Alabama ; Geol. Survey of Alabama," 3rd Edn., p. 240. Delameter, *Coal Age*, 1914, May 2nd. Drakeley, *Trans.* Inst. Min. Eng., 1918, 54, 518. Frazer and Yancey, *Trans.* Amer. Inst. Min. Eng., 1923, 69, 447.

† It may be pointed out, however, that any other specific gravity, considered to be more suitable than 1·35, may be selected.

To overcome the necessity of weighing the raw coal and refuse, the percentage of refuse may be written :

$$\frac{100\,(Fw - Ff)}{Fw - Fr} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (49)$$

and the quantitative efficiency is then equal to

$$\frac{100\,Fw\,(Ff - Fr)}{Ff\,(Fw - Fr)} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (50)$$

These expressions may be written in words,

$$\text{Qualitative Efficiency} = \frac{100\,(\text{Floats in washed coal} - \text{Floats in feed})}{100 - \text{Floats in feed}}$$

Quantitative Efficiency

$$= \frac{100\,(\text{Floats in washed coal})(\text{Floats in feed} - \text{Floats in refuse})}{(\text{Floats in feed})(\text{Floats in washed coal} - \text{Floats in refuse})}.$$

The calculation may be illustrated by two examples, with coals A and B. The results of float and sink tests are given in Table 155.

TABLE 155.—FLOAT AND SINK TESTS AT S.G. 1·35. COALS A AND B

	Coal A.		Coal B.	
	Floating per cent.	Sinking per cent.	Floating per cent.	Sinking per cent.
Raw coal . .	73·4	26·6	86·0	14·0
Washed coal .	96·6	3·4	97·5	2·5
Refuse . .	2.1	97·9	1·9	98·1

The efficiency of cleaning is as follows :—

Coal A.

$$\text{Qualitative} \quad . \quad . \quad . \quad . \quad \frac{100\,(96\cdot6 - 73\cdot4)}{100 - 73\cdot4} = 87\cdot2$$

$$\text{Quantitative} \quad . \quad . \quad . \quad . \quad \frac{100 \times 96\cdot6\,(73\cdot4 - 2\cdot1)}{73\cdot4\,(96\cdot6 - 2\cdot1)} = 99\cdot3$$

$$\text{General} \quad . \quad . \quad . \quad . \quad . \quad \frac{87\cdot2 \times 99\cdot3}{100} = 86\cdot6$$

Coal B.

Qualitative	$\frac{100\,(97{\cdot}5 - 86{\cdot}0)}{100 - 86{\cdot}0}$	$= 82{\cdot}1$
Quantitative	$\frac{100 \times 97{\cdot}5\,(86{\cdot}0 - 1{\cdot}9)}{86{\cdot}0\,(97{\cdot}5 - 1{\cdot}9)}$	$= 99{\cdot}7$
General	$\frac{82{\cdot}1 \times 99{\cdot}7}{100}$	$= 81{\cdot}9$

Frazer and Yancey combine Drakeley's quantitative efficiency formula with a part of Delameter's equations and give as a formula for general efficiency :—

$$\text{General efficiency} = \frac{\text{Yield of washed coal}}{\text{Yield of float coal}} \times \frac{\text{Raw coal ash} - \text{Washed coal ash}}{\text{Raw coal ash} - \text{Float coal ash}} \quad . \ (51)$$

By " yield of float coal," Frazer and Yancey mean the percentage of the raw coal which floats at some selected specific gravity. The yield of float coal is, in fact, the theoretical maximum yield of washed coal of the desired ash content. In the examples already given, this value would be 73·4 and 86·0 for coals A and B respectively. The yield of washed coal can be calculated from float and sink results and may be written :

$$\frac{100\,(Ff - Fr)}{Fw - Fr} \quad . \ . \ . \ . \ . \ . \ . \ . \ (52)$$

Substituting this value in Frazer and Yancey's formula, and writing Af, Aw and AFf for the percentage ash contents of the feed coal, the washed coal and the raw coal floatings, the formula becomes :

$$\text{General Efficiency} = \frac{100\,(Ff - Fr)}{Ff\,(Fw - Fr)} \times \frac{Af - Aw}{Af - AFf} \quad . \ (53)$$

The ash contents of coals A and B and their products are given in Table 156.

TABLE 156.—ASH CONTENTS. COALS A AND B

	Coal A per cent.	Coal B per cent.
Raw coal	19·5	12·5
Raw coal floats at S.G. 1·35	5·0	5·3
Raw coal sinks at S.G. 1·35	60·0	57·0
Washed coal	6·6	5·7
Refuse	59·2	56·0

· The General Efficiencies calculated from the Frazer and Yancy ormula are as follows:

$$\text{Coal A} \quad . \quad . \quad \frac{100\,(73{\cdot}4 - 2{\cdot}1)}{73{\cdot}4\,(96{\cdot}6 - 2{\cdot}1)} \times \frac{19{\cdot}5 - 6{\cdot}5}{19{\cdot}5 - 5{\cdot}0} = 92{\cdot}0$$

$$\text{Coal B} \quad . \quad . \quad \frac{100\,(86{\cdot}0 - 1{\cdot}9)}{86{\cdot}0\,(97{\cdot}5 - 1{\cdot}9} \times \frac{12{\cdot}5 - 5{\cdot}7}{12{\cdot}5 - 5{\cdot}3} = 97{\cdot}5$$

According to Drakeley's formula, coal A is washed more efficiently than coal B, whereas according to Frazer and Yancey's formula, coal B is the more efficiently washed. This latter conclusion would appear to be a more reasonable conclusion to draw, for, as shown by Table 155, the washed coal from coal A contains 3·4 per cent. of sinks, and the refuse 2·1 per cent. of floats, and the washed coal from coal B only 2·5 per cent. of sinks and the refuse only 1·9 per cent. of floats.

This illustrates one of the defects of the Drakeley formula, namely, that, with a coal containing large quantities of dirt, a comparatively higher qualitative efficiency is shown than when the raw coal is comparatively clean. For practical reasons, a high-ash coal is often easier to clean than a low-ash coal, and it is unfair to assign a higher figure for efficiency to an operation which is easier than to an operation which is more difficult. Drakeley's qualitative efficiency formula, whilst measuring the increased concentration of the floatable particles in the washed coal, takes no account of the quality of the unfloatable particles in the washed coal. Usually the "sinkings" found in washed coal are principally middlings particles, but Drakeley's qualitative efficiency formula does not take this into account and the washer is debited by an equal amount whether the "dirt" in the washed coal is middlings or pure shale. Similarly, the quantitative efficiency is reduced by the same amount whether the coal lost in the refuse consists of middlings particles or of "pure" coal.

Another objection to Drakeley's formula for quantitative efficiency is that, with any process which is even moderately efficient, the efficiency is about 99 per cent. It would appear, therefore, that when the quantitative efficiency is multiplied by the qualitative efficiency, there is insufficient distinction drawn by the quantitative term to permit the general formula to distinguish sufficiently between different operations. In the examples already given, the quantitative efficiencies were 99·3 and 99·7 respectively. If, in a given washery, 90 per cent. of the raw coal floated, and after washing the washed coal contained 100 per cent., of "floatings" (*i.e.*, no sinkings), but the refuse contained 10 per cent. of pure coal, the formula would give a quantitative efficiency of 98·8 per cent. Even if the refuse contained 20 per cent. of pure coal the quantitative efficiency would still be 97·2 per cent. In each case the qualitative efficiency would be 100 per cent. and the general efficiency would be

98·8 and 97·2 per cent. respectively. In the first case, however, 1·1 per cent. of the whole of the coal would be thrown away with the dirt, and in the second case 1·6 per cent. would be lost, and general efficiencies of nearly 100 per cent. would seem to be inappropriate.

There is another objection to Drakeley's formula, which applies in part also to Frazer and Yancey's formula, namely, that it assumes that there is no fracture of the middlings particles during washing. In general, British coals contain less middlings than coals from other sources, but they always contain a certain amount of middlings. The mechanical fracture of middlings during washing, and the disintegration often occurring when middlings particles are wetted, result in the conversion of some of the middlings into particles of "pure" coal and "pure" dirt. Consequently, the combined number of float particles in the washed coal and in the refuse may be greater than the number of float particles in the raw coal. Efficiencies of over 100 per cent. may then be experienced. This probably accounts for the example given by Grounds (*Proc.* S. Wales Inst. of Eng., 1927, 42, 545) of washing on H. H. tables which, he says, "savours of over 100 per cent. efficiency." Middlings particles, which before washing were sink particles, may have had a little shale chipped off them, or may have been broken into two particles, one high in ash which remained a sink particle, and one low in ash which became a float particle.

This objection to the Drakeley formula may be illustrated by an example.

The results of the float and sink analysis of a sample of raw coal fed to a Yorkshire nut coal washery is given in Table 157.

TABLE 157.—FLOAT AND SINK RESULTS. YORKSHIRE COAL

S.G.	Weight per cent. of Total.	Per cent. Ash in Fraction.	Cumulative Weight per cent.	Cumulative Ash Content per cent.
<1·3	48·2	2·64	48·2	2·64
1·3–1·4	19·1	7·33	67·3	3·97
1·4–1·5	10·4	15·68	77·7	5·54
1·5–1·6	4·0	28·25	81·7	6·65
1·6–1·8	8·7	43·50	90·4	10·19
>1·8	9·6	76·56	100·0	16·56

Suppose that the coal were being washed to recover, as washed coal, the material of specific gravity less than 1·5, and to reject that of specific gravity greater than 1·5, and suppose 3 per cent. of the S.G. 1·5 floats were lost in the refuse and 3 per cent of the S.G. 1·5 sinks passed into the washed coal. The yield of washed coal would

then be 0·97 × 77·7 + 0·03 × 22·3, or 76·0 per cent. of the raw coal fed, as against the theoretical yield of 77·7 per cent.

The ash content of the washed coal would depend upon whether the "sink" particles included in the washed coal consisted of heavy middlings (say, S.G. 1·6 to 1·8), or of shale (S.G. >1·8), and whether the "float" particles passing into the refuse consisted of "pure" coal (S.G. <1·3), or of middlings (say, S.G. 1·4 to 1·5). Four cases may be considered, and, according to the circumstances, the ash content of the washed coal would be as given in Table 158.

TABLE 158.—ASH CONTENTS OF WASHED COAL. HYPOTHETICAL CASES

S.G. of Material passing Accidentally into		Ash Content of Washed Coal per cent.*
Refuse.	Washed Coal.	
1·4–1·5	1·5–1·6	5·43
1·4–1·5	>1·8	5·85
<1·3	1·5–1·6	5·82
<1·3	>1·8	6·25

Under these conditions the ash content of the washed coal varies by nearly 1 per cent. In each case the washed coal contains

$$\frac{100\ (76\cdot04 - 0\cdot67)}{76\cdot04} = 99\cdot1$$ per cent. of particles floating at S.G. 1·5,

and the refuse, $\frac{100 \times 2\cdot33}{23\cdot96} = 9\cdot7$ per cent. of particles floating at

* The method of calculation is as follows :—

1·5 S.G. sinks passing into washed coal, 3 per cent. of total sinks, = 0·03 × 22·3 = 0·67 per cent. of raw coal.

1·5 S.G. floats passing into refuse, 3 per cent. of total floats, = 0·03 × 77·7 = 2·33 per cent. of raw coal.

First Example.—Washed coal consists of following fractions :—

S.G.	Weight.	Per cent. Ash.	Ash Contributed
< 1·3	48·20	2·64	1·272
1·3–1·4	19·10	7·33	1·400
1·4–1·5	8·07	15·68	1·265
1·5–1·6	0·67	28·25	0·189
Total .	76·04		4·126

$$\text{Per cent. ash in washed coal} = \frac{4\cdot126 \times 100}{76\cdot04} = 5\cdot43.$$

S.G. 1·5. The general efficiency according to the Drakeley formula would be :

$$\frac{100\,(99{\cdot}1 - 77{\cdot}7)}{100 - 77{\cdot}7} \times \frac{99{\cdot}1\,(77{\cdot}7 - 9{\cdot}7)}{77{\cdot}7\,(99{\cdot}1 - 9{\cdot}7)} = 94{\cdot}3$$

This would be the efficiency in each of the four cases considered, and no distinction whatever is drawn between them.

To be of any practical value, a formula for the efficiency of washing must draw at least as sharp a contrast between these four circumstances as do the ash content figures. This the Drakeley formula fails to do.

Frazer and Yancey's general efficiency formula is based partly upon the quantitative yield and partly upon the qualitative efficiency. Their qualitative term may be written :—

$$\frac{100\,(Ff - Fr)}{Ff\,(Fw - Fr)}$$

It will be seen that this value is identical with Drakeley's quantitative term except that the term Fw is omitted from the numerator. Drakeley's formula is, in effect, the yield of washed coal multiplied by the ratio of floats in the washed coal to floats in the raw coal. Frazer and Yancey's is the yield of washed coal divided by the floats in the raw coal.

Their qualitative term, which may be written :—

$$\frac{\text{Raw coal ash} - \text{Washed coal ash}}{\text{Raw coal ash} - \text{“ Floats in raw coal ” ash}}$$

is based upon ash contents and therefore draws some distinction between the loss of middlings or of coal in the refuse, and the recovery of middlings or of dirt in the washed coal. The four examples stated in Table 158 may be used to demonstrate this. The efficiencies calculated by the Frazer and Yancey formula are given in Table 159.

TABLE 159.—WASHING EFFICIENCIES. FRAZER AND YANCEY FORMULA

S.G. of Material passing Accidentally into		Ash Content of Washed Coal per cent.	Efficiency.
Refuse.	Washed Coal.		
1·4–1·5	1·5–1·6	5·43	86·7
1·4–1·5	>1·8	5·85	81·9
<1·3	1·5–1·6	5·82	82·1
<1·3	>1·8	6·25	79·8

An adequate differentiation between these four cases is shown by Frazer and Yancey's formula. The formula is unsatisfactory, however, under one set of conditions. The inaccurate operation of the washery may result in the loss with the refuse of particles which it is desired to pass into the washed coal (floats), there being, at the same time, no corresponding inclusion of sinks in the raw coal. In the examples considered, it was presupposed that 3 per cent. of the S.G. 1·5 floats passed into the refuse, and that 3 per cent. of the S.G. 1·5 sinks passed into the washed coal. Suppose, however, that 3 per cent. of the S.G. 1·5 floats passed into the refuse and that all the S.G. 1·5 sinks also passed into the refuse. The loss of float particles would amount to $0{\cdot}03 \times 77{\cdot}7 = 2{\cdot}33$ per cent. of the raw coal. If this loss consisted of particles of S.G. 1·4 to 1·5, the washed coal would consist of the following fractions :—

S.G.	Weight per cent. of Raw Coal.	Ash Content per cent.
$< 1{\cdot}3$	48·20	2·64
1·3–1·4	19·10	7·33
1·4–1·5	8·07	15·68

The yield of washed coal would, therefore, be 75·4 with a mean ash content of 5·22 per cent. The 24·6 per cent of refuse would contain—

$$\frac{2{\cdot}33 \times 100}{24{\cdot}6} = 9{\cdot}5$$

per cent. of floats at S.G. 1·5, and the efficiency of washing, according to Frazer and Yancey, would be—

$$\frac{100\,(77{\cdot}7 - 9{\cdot}5)}{77{\cdot}7\,(100 - 9{\cdot}5)} \times \frac{16{\cdot}56 - 5{\cdot}22}{16{\cdot}56 - 5{\cdot}54} = 101{\cdot}3$$

An efficiency of over 100 per cent. is therefore recorded when 2·3 per cent. of the raw coal is uselessly thrown away.

This example indicates the inadequacy of the Frazer and Yancey formula. The formula gives an excellent comparative value when the washer is producing a coal of higher ash content than is desired, but when the washer is producing too clean a coal, it is unduly credited for the improved quality of the coal, and insufficiently debited for the loss of output sustained. For it should be remembered that the object of washing is usually to obtain the maximum output of a coal of a given quality, and there is not usually any recompense for producing a coal of superior quality.

An additional objection to the Frazer and Yancey formula is that it necessitates a knowledge of the ash content of the raw coal.

The proportions of coal and dirt in the coal supplied to a washer, even when the supply is obtained from a large hopper, vary very considerably over short periods of time. The difficulties of collecting a representative example of the raw coal fed over a whole shift are, therefore, very great ; so great are they, especially when the washery deals with the larger sizes of coal, that it is almost impossible to obtain directly a true figure for the ash content of the raw coal. As this is required by the Frazer and Yancey formula, considerable errors may be introduced by using an inaccurate figure. The same objection applies to the Drakeley formula, for which float and sink analyses on the raw coal are required.

The fluctuations in the quality of the washed coal and the refuse are within much narrower limits, and are far less frequent than are the fluctuations in the relative proportions of coal and dirt in the raw coal. The washer acts as a stabiliser, and in all modern washeries provision is made to deal with a raw coal varying in quality and fed irregularly, the fluctuations being neutralised during the washing cycle. It is therefore much easier to obtain reliable samples of the washed coal and of the refuse produced during a shift than it is to obtain a representative sample of the raw coal.

For this reason it would be preferable that a formula for efficiency should be based solely upon the results of an examination of the products. By this means, not only is greater accuracy ensured, but the number of analyses required may be reduced.

A formula which would appear to meet the needs of washeries more satisfactorily than those already considered is as follows :—

$$\text{Efficiency} = \frac{\text{Ash content of raw coal floats}}{\text{Ash content of washed coal}} \times \frac{\text{Ash content of refuse}}{\text{Ash content of raw coal sinks.}}$$

It has been stated that the proportions of coal and dirt in the raw coal fed to a washery are liable to considerable variations, and this makes it difficult to determine the ash content or the percentages of float and sink in the raw coal. On the other hand, the ash contents of floats and sinks in the raw coal are subject to less variation, and these determinations are less liable to error.

Indeed, the coal itself and the dirt fed to any washery are of fairly uniform composition from hour to hour and from day to day, and average values for their ash contents may usually be taken. Such average values could be determined from figures over a period of several months, and if the ratio

$$\frac{\text{Ash content of floats in raw coal}}{\text{Ash content of sinks in raw coal}}$$

were found to have a fairly constant value (K), the suggested formula for efficiency could be written simply :—

$$\text{Efficiency} = \frac{K \times \text{ash content of refuse}}{\text{Ash content of washed coal}}.$$

To make the theoretical maximum value 100, for convenience, K could be multiplied by 100, and the formula could then be stated :—

$$\text{Efficiency} = \frac{C \times \text{ash content of refuse}}{\text{Ash content of washed coal}},$$

where C is a constant determined from the average float and sink analysis of the raw coal over a period of time, and is equal to

$$\frac{100 \times \text{ash content of floats in raw coal}}{\text{Ash content of sinks in raw coal}}.$$

This formula has several advantages over others. It requires only two analyses, it is simple to calculate and easy to remember. It takes account automatically of the quality of the material going astray (*i.e.*, coal or middlings into the refuse ; pure dirt or middlings into the washed coal) and of the yield of washed coal. Furthermore, it does not require sampling of the raw coal (except periodically to check the value of C), and the ash content of the saleable coal being usually determined as a matter of routine, requires only one extra daily analysis.

The examples already considered in connection with the Drakeley and Frazer and Yancey formulæ (Table 158) may be employed to demonstrate the use of the suggested formula. In all these examples it was presupposed that the washery was operated to make a separation of the material lighter than a specific gravity of 1·5 from the heavier material. The ash content of the S.G. 1·5 float is 5·54 per cent., and of the S.G. 1·5 sinks 55·00 per cent. In these circumstances, $C = \frac{100 \times 5{\cdot}54}{55{\cdot}00} = 10{\cdot}07$.

The ash contents of the products and the efficiencies are given in Table 160.

TABLE 160.—CONSTITUENTS OF PRODUCTS AND EFFICIENCIES

S.G. of Material passing Accidentally into		Ash Content per cent.		Efficiency.
Refuse.	Washed Coal.	Washed Coal.	Refuse.	
1·4–1·5	1·5–1·6	5·43	51·92	96·3
1·4–1·5	> 1·8	5·85	50·57	87·1
< 1·3	1·5–1·6	5·82	50·65	87·6
< 1·3	> 1·8	6·25	49·30	79·4

The suggested formula places these four examples in the same order of efficiency as the Frazer and Yancey formula, but the variation in efficiency is greater, the range being 17 units instead of ·7 units.

The new formula also places the efficiency of washing coals A and B (Tables 155 and 156) in the same order as does the Frazer and Yancey formula, and in the reverse order to that suggested by Drakeley's formula.

Formula of	Efficiency. Coal A.		Coal B.
Drakeley . . .	86·6	..	81·9
Frazer and Yancey .	92·0	..	97·5
Chapman and Mott .	75·9	..	91·4

The wider range between the efficiencies suggested by the new formula is again shown. It will be noticed that, according to the formula employed, the efficiency varies in any one case. Thus, for coal A, the efficiencies are 86·6, 92·0 and 75·9 respectively. This is of no moment, for the figures have no absolute value. They are only relative, and it is doubtful if any of them have any real value otherwise than as a means of comparing daily (or weekly) operations.

In using the formula, it is necessary first to decide at what specific gravity the separation of the raw coal into coal and dirt can be most economically attempted. If the colliery management decided that the market demand would make it more profitable to effect a separation at some specific gravity other than 1·5 (as in the examples given) the value of the constant, C, would be different. With separation at specific gravities of 1·4, 1·6 or 1·8, the floats in the raw coal would have ash contents of 3·97, 6·65 or 10·19 per cent. and the ash content of the sinks in the raw coal would be 42·50, 60·81 or 76·56 respectively. The constant, C, would then have the value 9·34, 10·93 or 13·31.

Under present conditions there is little incentive for a colliery company to reduce the ash content of its coal below a certain amount. If coal containing 7 per cent. of ash is readily saleable and coal with only 6½ per cent. of ash commands no higher price in the market, a loss of saleable material is encountered if the coal is cleaned to below 7 per cent. of ash. The formula suggested credits the washery if the coal is washed "too clean." Thus, if the separation is made at S.G. 1·5, the floats contain 5·54 per cent. of ash; but if the coal is washed to, say, 5·40 per cent. of ash, the calculated efficiency is higher than present market conditions justify. For this reason, in using the formula, it would appear to be more satisfactory to place a minimum value on the ash content of the washed coal.

It has been assumed in Table 160 that separation at S.G. 1·50 is attempted In the first example some of the S.G. 1·4 to 1·5 middlings passed into the refuse and some of the S.G. 1·5 to 1·6 middlings

passed into the washed coal. The washed coal then contained 5·43 per cent. of ash, whereas the S.G. 1·5 floats in the raw coal contained 5·54 per cent of ash. In this example, we suggest that a minimum value of 5·54 for the ash content of the washed coal should be used in calculating the efficiency. The efficiency would then be

$$\frac{10{\cdot}07 \times 51{\cdot}92}{5{\cdot}54} = 94{\cdot}4$$

instead of

$$\frac{10{\cdot}07 \times 51{\cdot}92}{5{\cdot}43} = 96{\cdot}3.$$

It should be stated that, when the coal is not washed "too clean," the calculated efficiency seems to provide a satisfactory means of comparing the day to day operation of the washery. In common with all efficiency formula, however, it suffers from the defect that no relation to market price is incorporated. In this connection, Chapter XXXIII. should be consulted.

CHAPTER XXXII

THE ADVANTAGES OF CLEAN COAL IN INDUSTRY

FOR whatever purpose coal or coke is used, it is to the advantage of the consumer that the fuel should contain the minimum amount of ash. Incombustible material in the fuel reduces its gross calorific value, increases the weight that must be handled and transported, gives rise to difficulties of combustion which render the heat less easily available, occasions losses of combustible material, and involves further expense in its disposal. Its disadvantages are not confined to the consumer; they are also inflicted upon the general public, because ash in coal increases the production of smoke and results in the discharge of fine dust from chimney stacks, especially from pulverised fuel boilers.

If it be assumed that, by cleaning, the ash content of the coal could be reduced, on the average, by 5 per cent., from, say, 10 per cent. to 5 per cent., it can be calculated that about £2,500,000 is spent per annum on the transport of useless material; that, in Great Britain, 5,000,000 tons of removable incombustible matter is charged per annum to boiler fires. These figures have little real significance, because they are based on rather wide assumptions, but their magnitude is, perhaps, rather greater than would be casually imagined.

Although it is generally agreed that the consumer would be willing to pay for the removal of this incombustible material in proportion as the heating value of the residual fuel was increased, the full extent of the benefits that would accrue to the consumer are not, perhaps, adequately realised. The additional value of the purer fuel is, for almost all purposes, considerably greater than is suggested by a comparison of the calorific values of the two fuels. This may be demonstrated by describing briefly the harmful effect of the ash in coal used for various purposes and the advantages which a purer coal, as obtained by coal cleaning, would bring in its train.

THE CARBONISATION INDUSTRIES

In the manufacture of by-product coke there are certain difficulties arising because of the presence of incombustible matter in the coal. For example, the presence of chlorides is very harmful to the oven walls, except when they are made of silica, and the presence of iron tends to reduce the ammonia yield. The principal disadvantage of dirt in the coal, however, is the production of breeze in the coke. Breeze always contains a higher percentage of ash than the larger

coke, and it is clear that it results from the presence of slurry in the coal. The slurry usually contains at least 20 per cent. of ash and 30 per cent. of water, and includes much fusain and clay slimes, which prevent the agglomeration of the slurry into a hard coke. The weak product which results is easily disintegrated and accumulates in the breeze collected at the coke-oven plant. In certain Durham cokes it has been observed that thin flat pieces of shale, which have not been removed in washing the coal, cause fractures in the coke, thus making it more liable to break and form breeze. Fine crushing would reduce the size of the shale particles, which would then be more evenly admixed with the coal. This is, however, the wrong method to use, for it is certain that most of these particles could be removed by washing in a suitable plant, and most desirable that they should be removed.

Unscreened Durham gas coal is used to a very large extent in the gas industry, particularly in the South of England. A typical Durham gas coal contains from 10 to 11 per cent. of ash, yielding a coke containing from 14 to 15 per cent. of ash. Such material is known to be much less efficient in use than a cleaner coke, and, with modern developments in coal cleaning, there is no excuse for its continued use.

In modern gasworks carbonisation a portion of the coke is gasified in producers and the remainder is sold as a domestic fuel. The removal of the clinker from the producers and its disposal is an expensive charge on the gasworks, and, in large towns, may amount to four or five shillings per ton of ashes. The incombustible mineral matter is, moreover, a source of trouble and expense in the operation of the producer. The greater the ash content of the coke, the more frequently is it necessary to clean the fires and remove clinker, and the gas-making capacity of the producer is proportionately reduced. Further disadvantages of high-ash fuels are the increased amount of combustible matter lost in the ash and the rapid increase in the resistance to the flow of air and steam through the fuel bed between clinkering periods. The channelling which occurs when the resistance increases occasions additional difficulties because it causes fine solid particles to be carried forward with the gas. Similar difficulties are experienced when coke is used for the manufacture of producer gas for metallurgical purposes.

Moreover, in the gas retort, the yield of gas per ton of coal is reduced by the presence of dirt in the coal charged and the sulphur content of the gas may be increased.

DOMESTIC FUEL

The principal objection to gasworks coke as a domestic fuel is that it is usually dirty. At present, the importance of sizing coke for the domestic consumer is being realised, but it is not fully realised that the domestic consumer is usually far more likely to purchase a coal

because it burns in an open grate without leaving large quantities of dust and ashes behind, than because it ensures the maintenance of a clear atmosphere above his chimney top. Yet gas companies recommend their coke for its smokeless qualities. They would probably attract more customers if they were able to recommend it for its ashless qualities, and such customers would be prepared to pay a higher price because it reduced domestic labour and expense.

Apart from the labour of dusting the room after clearing the ashes, and the damage to carpets and furnishings by the fine gritty particles of ash, there is a definite reduction of thermal efficiency of the fire when the coke contains quantities of incombustible matter. Bligh and Hodsman, for example (*Journ.* Soc. Chem. Ind., 1927, 46, 92T), found that the radiant efficiency of a domestic coke fire was 29·3 per cent. with a coke containing only 1·5 per cent. of ash, whereas with a similar coke containing 5·5 per cent. of ash the radiant efficiency was only 23·9 per cent. An increase of 5·4 per cent. in the utilisation of the potential heat of the coke was obtained by reducing the ash content of the coke by only 4 per cent.

Bligh and Hodsman describing the tests state that: "The superiority of the fuel was obvious, apart from any measurement, both as regards heating efficiency and appearance." They attribute the decreased efficiency with the coke of higher ash content to a film of incombustible material formed over the surface of the particles.

The objections to the ash contained by domestic coke apply also to the incombustible matter present when coal itself is used as a domestic fuel. When coal is used, however, the bulk of the ash occurs as "fixed" ash in the coal, though large pieces of free dirt are often found and the removal of these pieces irritates a householder who often has to pay a high price for the coal.

BUNKER COAL

The chief objections to ash in the coal used for bunkering are those common to all coal used for boiler firing, and these objections are described at length later. There are, however, two additional objections. Firstly, uncleaned coal contains a greater proportion of pyrites than cleaned coal, and although pyrites may not, in the majority of cases, be the primary cause of spontaneous combustion, there is little doubt that it is, or can be, a contributory factor.

The second additional objection is the extra cost of bunkering and the loss of cargo space. If 100 tons of coal containing 10 per cent. of ash are bunkered at a cost of 1*s*. per ton, 10*s*. is spent on loading useless material and an extra 10 tons of dead weight are carried, which, at a freight rate of 1*s*. 9*d*. per ton per day, is equivalent to a loss of earning power of 17*s*. 6*d*. per day, or about £250 per annum. A reduction of the ash from 10 to 5 per cent. would therefore effect a considerable saving.

RAILWAY FUEL

The railway companies have for many years realised the value of a coal with a relatively low ash content, because of its higher calorific value and the increased rates of evaporation per square foot of grate area. For these reasons they have always been willing to pay a higher price for lump coal, which, besides being easier to stack and less liable to spontaneous combustion, almost always has a lower ash content than the smaller sizes. Before the coal is used for locomotive firing, however, it is broken up and usually moistened, so that, except for the greater ease and safety of storage, small and moist coal, provided it were clean, would be equally suitable.

THE VALUE OF CLEAN COAL FOR STEAM GENERATION IN BOILERS

There are seven principal causes of thermal losses in the generation of steam in any kind of boiler other than losses due to bad design or poor operation. These are as follows :—

(*a*) Unburnt carbon in the ashes.
(*b*) Unburnt carbon in the flue gases.
(*c*) Combustible gas in the flue gases.
(*d*) The sensible heat of the flue gases.
(*e*) The sensible heat of the ashes.
(*f*) The heat required to evaporate the moisture in the coal.
(*g*) Radiation from the body of the boiler plant.

The use of coal of high ash content results in a greater loss of efficiency than is experienced when using a coal with a low ash content for each of the first five causes.

It is obvious that a coal with a high ash content will yield a greater amount of ashes than a cleaner coal, and that the loss of sensible heat, and of combustible matter, in the ashes will therefore be greater. But it is also true that a high ash content leads to greater losses in the flue gases.

When coal is burnt on a grate, the irregularities of the depth of the fuel bed cause too much air to be passed where the bed is thin and too little where the bed is thick. In order to prevent the formation of smoke the air supply must be regulated to meet the needs of the thicker parts of the bed, with the result that an excess of air passes through the bed where it is thin. Some of this excess of air is utilised in the space above that occupied by the solid fuel for the combustion of the volatile hydrocarbons distilled from the coal, but the greater part of it appears in the flue gases, carrying away sensible heat with it.

It is necessary to pass some excess air through the fuel bed in order to prevent a high loss of fuel in the flue gases, either as carbon monoxide or as smoke. But it is well known that if the quantity of excess air passed is too great, smoke formation is enhanced, because

the air causes a local reduction in the temperature above the fuel bed and opposes complete combustion. The gases passing over the fuel bed are then insufficiently heated and are imperfectly burned. The time available for their proper combustion is reduced, because the greater is the amount of excess air, the more rapidly do the gases above the fuel bed pass into the flues.

The speed with which the air passes through the thinner portions of the fuel bed causes small particles of solid fuel to be lifted from the bed, and these combustible particles pass into the flues without being burned. The more rapidly the air passes through the channels in the bed, the greater is the loss from this source of unburnt carbon in the flue gases. The solid particles blown from the bed by the draught of air also tend to stick to the crown of the furnace, and, with some coals, the ash fuses at the high temperature of the crown and the refractory lining of the furnace is eroded.

These losses of heating power in boilers are common to all boilers, whether fired by hand, by a mechanical stoker, or by pulverised fuel. They are greatest in a hand-fired boiler because of the greater irregularities of the bed, and because of the admission of cold excess air over the fire whenever the door is opened for firing or for clinkering. With a mechanical stoker, more regular conditions of the fire are ensured, but channelling still requires the admission of about 75 per cent. of excess air, and the forced draught causes appreciable losses of small solid combustible particles (other than as smoke) in the flue gases. In a pulverised-fuel furnace all the losses of combustible matter and of sensible heat are minimised.

The losses experienced in all three types of boiler are increased considerably if the boiler is called upon to perform work more than its normal rated duty. All boilers, in good condition and properly fired, can be satisfactory at a low, steady load, but steady conditions are seldom encountered, and provision must always be made for sudden and irregular demands for steam. The efficiency falls very rapidly in an overloaded boiler fired by hand until, when the boiler carries double its normal load, the efficiency may have fallen from 70 to 40 per cent. The efficiency of a mechanical stoker also falls rapidly at overloads, and stand-by boilers have usually to be ready to meet the peak load in a power station. A pulverised-fuel furnace can, however, be made to meet sudden and increased loads with only a relatively small fall of efficiency.

The effect of using a high-ash coal instead of a low-ash coal may now be examined. A hand-fired boiler, which is the least efficient of the three types mentioned, is affected to the greatest extent. More frequent firing is required, and this necessitates the more frequent opening of the doors. The excess air admitted over the fire cools the burning gases, producing smoke, and is itself heated, causing a loss of sensible heat. It is necessary to rake the fire and to clinker at shorter intervals, again allowing the entry of cold excess air above the fire, and causing a much greater amount of combustible material

to fall through the bars and be lost. It is also necessary to work with a greater depth of fuel bed. A thick bed of fuel aggravates the difficulties of maintaining an even bed and greatly increases channelling, and the losses of sensible heat and combustible matter caused thereby.

Mechanical stokers are unable to deal with coals which clinker easily. With clinkering coals, sticking in the grate interferes seriously with the draught. This difficulty can be met by steam injection, but it is apt to be considerably more serious if a high-ash coal is used. In any event, an increased amount of ash in the fire increases the resistance to the passage of the air and deadens the fire, renders a break-through of the air in a weak part of the bed much more likely, and causes a greater loss of combustible matter in the ashes. If the boiler is required to meet a sudden demand, the fire is less able to respond to an increased rate of feed, and the percentage loss of combustible matter in the ashes increases rapidly. Not only does a high-ash coal increase the losses of sensible heat due to the passage of excess air, but it also increases the loss of combustible particles in the flue gases, and in the ashes. Moreover, it materially reduces the flexibility of the boiler.

It has frequently been stated that pulverised-fuel firing offers an outlet for the use of accumulations of low-grade fuel. Low-grade fuels certainly can be and are being used in pulverised fuel installations, but where the coal could be cleaned before use, the efficiency would be greater and the cost of cleaning would be a profitable expense.

When a high-ash coal is used, the cost of drying is correspondingly increased, and the cost of grinding and the wear and tear of the pulveriser are disproportionately increased because the shale is harder to grind than the coal. Furthermore, the mineral matter in the dust entering the combustion chamber must be heated to the flame temperature without contributing anything to the heating and the incombustible dust must be discharged from the furnace. Generally the flue gases carry with them large quantities of incombustible dust and discharge them from the stack. Much of the finely-divided dust so discharged passes into the upper atmosphere and is not brought down except by rain, but a part of it, at least, falls to the ground. The more rapidly the practice of firing boilers with pulverised fuel spreads, the more numerous will be the complaints against this nuisance, and the probable development of pulverised fuel firing is likely to increase the number of complaints and the number of interests behind them.

The efficiency of a pulverised-fuel boiler is impaired in several ways by employing a coal with a high content of mineral matter. The incombustible dust is apt to collect on the water tubes and in the flues. These collections must be removed periodically. This source of loss is magnified if the ash is readily fusible, and coals with even a slightly fusible ash are also expensive to use because of the severe

erosion of the walls. Indeed, the wear and tear on refractory walls has been responsible for a re-design of pulverised-fuel boilers. In the latest designs the walls are of steel, but the substitution of steel fin-tube walls for air-cooled walls in order to remove the difficulties caused by the ash in the coal is not without its own difficulties. Coals with a high ash content are more likely to lead to the troubles caused by ash fusion and by erosion than are those with a low ash content. The particles exist in the combustion chamber as finite units, and the individual particles are not all of the same composition, some of them being more fusible than others. The amount of trouble to be anticipated from the harmful effect of the ash is roughly proportional to the amount of ash present. Furthermore, if x lb. of a coal containing 5 per cent. of ash is required to give a certain rate of evaporation, the amount of ash produced is $0{\cdot}05x$. The substitution of a coal with 20 per cent. of ash requires a minimum feed of $\frac{100-5}{100-20}x$ to obtain the same heating effect, and the amount of ash produced is in the proportion $\frac{20}{100} \times \frac{95x}{80}$ or 24 to 5 instead of simply 20 to 5.

The use of a coal containing large quantities of incombustible matter requires an increase in the rate of feed in order to maintain the required temperature and the same rate of evaporation, and, in effect, reduces the size of the combustion chamber, partly because of the more rapid entry and passage of the particles, but also because, in the latter stages of the combustion, the solid combustible particles are shielded from the radiant heat of the walls (which accelerates combustion) by the incombustible particles which are interposed. The ability of a boiler fired by pulverised fuel to meet rapidly varying conditions is its greatest advantage over a boiler fired by other methods. But if the coal has a high ash content, and the feed is forced in order to meet a sudden overload, the difficulties attending too high a velocity of feed will be accentuated, the liability of troubles due to fusion of the ash will be increased, the complete combustion of the solid particles remaining after distillation and combustion of the volatile matter will be rendered more difficult, and the troubles attending the removal of dust from the flue gases will be intensified.

Obviously, then, the use of a coal containing considerable quantities of incombustible material is largely neutralising the principal advantage which pulverised-fuel firing can offer over other systems of firing.

The quantitative estimation of the losses of efficiency and flexibility of all kinds of boiler by reason of the incombustible mineral matter contained in the fuel used is hampered by lack of available test figures. In 1917 the American National Research Council issued a pamphlet prepared by the J. G. White Engineering Corporation, in which the losses of boiler efficiency due to the ash content of

the coal were assessed. The number of pounds of coal required per boiler horsepower-hour with different qualities of fuel was stated to be as follows :—

Ash Content of Coal per cent.	Coal required per Boiler h.p. hour. Lb.
4	3
8	3·2
12	3·5
15	3·9
18	4·4
21	5·4

Similarly, on account of the reduced boiler capacity when burning a high-ash coal, a larger number of boilers are required. Thus, to generate 300,000 lb. of steam per hour, equivalent to a peak load of 15,000 to 20,000 kw., the following numbers of boilers are required :—

Ash Content of Coal per cent.	No. of Boilers required.
4	8
8	9
12	11
15	15
18	18
21	20

These relative figures are based on the practical experiences of a very large industrial company, and in no uncertain way indicate the much greater operating expense when a high-ash coal is used instead of a low-ash one.

The following figures are estimates of the losses of efficiency in hand-fired boiler practice when the fuel used is bituminous coal containing 4, 10 and 20 per cent. of ash respectively, and when the conditions of firing may be regarded as reasonably good.

The calculations have been made so that the miscellaneous losses of efficiency have been made equal to 7·5 per cent. in each case. These losses include the loss due to the moisture in the coal, radiation and other losses, so that the figures apply to coals with equal moisture contents, and under similar conditions of firing. The estimated proportionate losses with coals differing only in ash content are given in Table 161.

On the figures given in Table 161, the operating efficiency with a coal containing only 4 per cent. of ash is 11·5 per cent. greater than with a coal containing 10 per cent. of ash, and 24·6 per cent. greater than with a coal containing 20 per cent. of ash. These figures, however, do not represent the full advantage to be gained by the use of a coal low in ash. They take no account of the greater flexibility of the boiler, the smaller number of boilers required for a given demand, and the greatly decreased cost of ash-handling and disposal.

TABLE 161.—AVERAGE HEAT LOSSES IN HAND-FIRED BOILERS. HEAT LOSSES EXPRESSED AS PERCENTAGES OF HEAT AVAILABLE

Cause of Loss.	Ash Content of Coal.		
	4.	10.	20.
Loss of combustible matter :—			
(a) As carbon in the ashes	0·6	4·8	10·0
(b) As carbon monoxide and smoke in the flue gases	0·9	1·9	3·5
(c) As solid combustible particles in the flue gases	1·0	1·5	2·0
Loss of sensible heat—			
(a) In the flue gases	12·2	17·8	23·0
(b) In the ashes	0·1	0·3	0·9
Other losses, radiation, moisture in flue gases, etc.	7·5	7·5	7·5
Thermal efficiencies	77·7	66·2	53·1
Totals	100·0	100·0	100·0

Suppose slack containing 10 per cent. of ash were available at the pithead at a cost of P shillings per ton and that the coal were transported by rail for A miles at 1*d*. per ton-mile (the approximate present cost). Suppose, also, that the cost of ash handling and disposal were B shillings per ton and that C per cent. of the coal were converted into ashes. The cost of 66·2 units of efficiency would be

$$P + A/12 + BC/100.$$

Or, one unit of efficiency would cost

$$\frac{P + 0{\cdot}083A + 0{\cdot}01BC}{66{\cdot}2} \text{ shillings.}$$

For an average location, A may be given an arbitrary value of 40, and the cost of ash handling and disposal may be reckoned as 5*s*. per ton. For the purpose of calculating the relative values of coals containing 4, 10 or 20 per cent. of ash, a basic value of 10*s*. may be taken for a ton of slack containing 10 per cent. of ash at the pithead.

The cost per unit of efficiency, with a 10 per cent. ash coal, is therefore

$$\frac{10 + 3{\cdot}33 + 0{\cdot}105 \times 5}{66{\cdot}2} = 0{\cdot}2097 \text{ shillings}$$

the percentage of ashes formed being 10·5.

With a 4 per cent. ash coal, 77·2 units of efficiency are obtained and 4·02 per cent. of ashes are made. If the value of a unit of efficiency be taken as 0·2097 shillings,

$$\frac{P + 3{\cdot}33 + 0{\cdot}0402 \times 5}{77{\cdot}2} = 0{\cdot}2097 \text{ shillings.}$$

Whence, P = 12·66 shillings.

Similarly, for a 20 per cent. ash coal, with an efficiency of 53·1 per cent. and producing 22·22 per cent. of ashes,

$$\frac{P + 3{\cdot}33 + 0{\cdot}2222 \times 5}{53{\cdot}1} = 0{\cdot}2097 \text{ shillings.}$$

Whence, P = 6·69 shillings.

On this basis, if a ton of coal containing 10 per cent. of ash is worth 10*s*. at the pit, a coal with 4 per cent. of ash is worth 12*s*. 8*d*., and a coal with 20 per cent. is worth only 6*s*. 8*d*. These values, however, make no allowance for the greater flexibility and capacity of the boiler when the coal with the lower ash content is used.

These figures assume, as previously stated, an equal moisture content of the coals. Unfortunately, washed coals are frequently badly drained and the advantage of low ash content is often, in part, neutralised by the higher losses involved in evaporating the moisture in the coal.

The relative losses of heating value when coals with different ash contents are burned on mechanical stokers are more difficult to assess because of the differences between different makes of stoker, especially in respect of the losses of combustible matter in the ashes (in distinction to the clinker) produced by the mechanical disintegration of the fuel as it travels forward with the grate.

According to Bone ("Coal and its Scientific Uses," London, 1918, p. 195), "There seems to be a consensus of opinion amongst competent judges that, from the point of view of fuel economy, the advantages of mechanical stoking over hand firing are greater the lower the grade of fuel employed. Indeed, mechanical stokers tend to diminish the natural difference between a low-grade fuel in a degree which increases with the extent of such differences."

In this statement, Bone is referring principally to coals of varying rank (bituminous compared with sub-bituminous, etc.) rather than to coals with different ash content, but the statement seems to be also generally applicable to coals of similar rank, but varying in ash content up to about 15 per cent. Above about 15 per cent., the heat losses in operating a mechanical stoker plant increase rapidly with increasing ash content of the fuel. Consequently, although it is distinctly advantageous to use a good quality fuel on a hand-fired boiler, the advantage of a coal of low ash content over a lower quality fuel is less apparent on mechanical stokers. In general, with mechanical stoking, there is a higher loss of sensible heat in the flue gases than with hand firing, but there is considerably less loss of combustible matter in the flue gases as carbon monoxide and as smoke.

The following figures (Table 162), published by Patterson (*Chem. and Ind.*, 1923, 42, 904), which compare the amounts and carbon contents of the ashes produced in fifteen different large mechanical stoker plants, are of interest.

TABLE 162.—ASHES PRODUCED IN FIFTEEN MECHANICAL STOKER PLANTS

Coal.	Volatile Matter per cent.	"Fixed Carbon" per cent.	Ash per cent.	Carbon in Ashes per cent.	Carbon in Ashes as per cent. Coal.	Size of Coal.
A	28·90	51·16	16·34	17·24	3·34	½ in. slack.
B	27·96	50·94	17·38	11·61	2·40	Dust to ½ in.
C	25·38	50·54	18·88	22·84	5·50	Small slack.
D	24·98	52·68	20·38	13·32	3·00	¾ in. slack.
E	30·84	57·30	9·70	11·22	1·20	1 in. slack.
F	24·86	47·60	25·40	30·44	11·00	Dust to ½ in.
G	28·18	51·62	14·18	15·88	2·60	Clean pea slack.
H	26·00	46·36	23·68	17·64	5·00	½ in. slack.
K	26·06	48·04	22·00	13·40	3·40	½ in. slack.
L	23·49	61·49	25·02	21·04	6·60	Dust to 1 in.
M	33·47	52·73	13·80	5·20	0·75	1 in. nuts.
N	7·53	69·86	22·61	31·00	9·80	Fine anthracite slack.
O	20·42	69·87	9·71	9·30	0·97	Clean 1 in. slack.
P	26·06	60·82	11·00	25·20	3·77	½ in. slack.
Q	26·80	43·44	28·16	40·48	19·10	Very fine slack.

If these figures be grouped to compare the amounts of combustible matter lost from coals of varying ash content, there is, with a few exceptions, a rapid increase in the losses with an increase in the ash

TABLE 163.—LOSS OF COMBUSTIBLE MATTER WITH COALS OF INCREASING ASH CONTENT

Coals.	Range of Ash Content per cent.	Mean Loss of Carbon as per cent. of Coal.	Mean Loss of Carbon as per cent. of Fixed Carbon in Raw Coal.
E, O, P, M, G	Under 15	1·86	3·22
A, B, C	15 to 20	3·75	5·70
D, K, N, H, L	20 to 25	5·56	9·68
F, Q	Over 25	15·55	33·55

content of the raw coal. The losses are more striking if the loss of carbon is expressed as a percentage of the amount of " fixed carbon " in the raw fuel. The comparisons are made in Table 163.

The large increase in the loss of combustible matter with increase in ash content of the raw fuel is evident from these figures. The amount of loss is more significant if the loss is expressed as a percentage of the amount of " fixed carbon " in the ash-free coal. The " fixed carbon " content of the ash-free coal may be calculated from Patterson's figures, and the mean loss of carbon on this basis for coals E, O, P, M and G, which have a mean ash content of 11·7 per cent. and a mean " fixed carbon " content of 58·47 per cent., is

$$\frac{3{\cdot}22 \times (100 - 11{\cdot}7)}{58{\cdot}47}$$

The carbon losses so calculated are :—

Ash Content of Coal.	Loss of Carbon as per cent. Fixed Carbon in Ash-free Coal.
11·7	4·9
17·5	9·2
22·7	13·4
28·0	53·1

From the figures in Table 163 it appears that the loss of carbon in the ashes does not begin to rise rapidly until the coal contains about 20 per cent. of ash, and, similarly, other losses of heating value are relatively less serious with coals containing below 15 to 20 per cent. of ash than with coals containing more than this amount.

The advantage of using clean coal in mechanical stokers is, therefore, largely dependent upon the relative costs of cleaned and uncleaned slacks, considered in conjunction with the amount of moisture left in the coal cleaned by wet washing, the distance over which the coal must be transported for use, and the cost of ash-handling.

The value of a coal for steam generation at a large electricity generating station is usually assessed by trial, but if this is impossible reference is often made to a chart, giving the number of B.Th.U. available per penny. This is the practice at the Electric Supply Department of the Corporation of the City of Sheffield. The calorific value of the dry coal is multiplied by a utility factor (based on practical experience) to give the number of B.Th.U. probably available. This is divided by the delivered price, plus the estimated cost of ash disposal. The formula is given by Miles,* (whose book should be consulted for further information) as follows :—

$$\frac{\text{B.Th.U. per penny}}{\text{coal as received}} = \frac{\text{B.Th.U.} \times \text{S.F.} \times 2{,}240}{\text{Price in pence} + \text{ash disposal cost}}$$

* " The Chemistry of the Power Plant." London, 1925.

TABLE 164.—VALUATION OF COALS FOR STEAM GENERATION

B.Th.U. on Dry Coal.	Utility Factor.	B.Th.U. available Dry Coal.	Per cent. Ash in Dry Coal.	Per cent. Ashes Produced.	Cost of Ash Disposal, pence per Ton of Coal.	Available B.Th.U. (thousands) per penny. Price per ton Delivered.						
						5s.	7s.	9s.	11s.	13s.	15s.	20s.
14,680	0·894	13,120	2·0	3·2	1·9	474·0	342·0	267·3	219·2	186·1	161·4	121·4
14,600	0·891	13,020	2·6	3·6	2·2	470·5	339·0	265·0	217·6	184·6	160·3	120·6
14,200	0·877	12,460	5·3	5·7	3·4	439·4	318·9	250·3	206·0	175·0	152·1	114·7
13,800	0·860	11,880	8·0	8·4	5·0	409·5	299·0	235·5	194·3	165·3	143·8	108·6
13,400	0·841	11,280	10·5	11·5	6·9	377·0	277·6	219·7	181·8	155·0	135·1	102·3
13,000	0·818	10,640	13·2	14·9	8·9	345·3	256·6	203·6	169·0	144·4	126·1	95·7
12,600	0·791	9,990	15·9	18·7	11·2	314·2	234·8	187·5	156·1	133·6	116·8	88·9
12,200	0·758	9,250	18·7	22·5	13·5	281·8	212·4	170·5	142·3	122·2	107·0	81·7
11,800	0·722	8,530	21·2	26·9	16·1	251·4	191·0	154·0	129·0	111·1	97·4	74·6
11,400	0·682	7,770	23·8	31·3	18·8	220·4	169·0	137·1	115·3	99·4	87·4	67·2
11,000	0·634	6,970	26·5	36·2	21·7	191·5	148·0	120·5	101·1	87·9	77·5	59·7
10,500	0·559	5,870	21·8	42·5	25·5	153·8	120·0	98·5	83·5	72·4	64·0	49·5
10,000	0·470	4,700	33·0	48·7	29·2	118·2	93·1	76·8	65·4	56·9	50·4	39·1
9,500	0·348	3,300	36·5	57·0	34·2	78·6	62·6	52·0	44·5	38·9	34·5	27·0
9,000	0·193	1,740	40·0	66·6	40·0	38·9	31·4	26·3	22·7	19·9	17·7	13·9

$$\text{Or, B.Th.U. per penny dry coal} = \frac{\text{B.Th.U. (dry)} \times \text{S.F.} \times 2{,}240}{\text{Price in pence} + \text{ash disposal cost}}.$$

S.F., the suitability or utility factor, is based essentially on practice, and relates the number of B.Th.U. in the coal to the number of B.Th.U. that can be obtained from it by combustion in a mechanically-stoked boiler. The approximate calorific value of the dry coal, and the amount of ash that will be produced in operation, can be calculated (on the basis of previous experience) from the ash content of the dry coal; pure, dry, ashless coal being assumed to have a calorific value of 15,000 B.Th U. per lb. The suitability factor makes allowance for the losses of heat that will be sustained as sensible heat in the flue gases, combustible matter in the ashes and the gaseous products of combustion and for other losses.

The figures given in Table 164 have been selected from a chart compiled on this basis, the suitability or utility factors being those found to obtain at the Sheffield Electric Supply Department stations, and differing only slightly from those given by Miles.

To obtain a yield of a given number of B.Th.U. per 1*d*. of fuel

TABLE 165.—PERMISSIBLE ASH CONTENT OF COAL TO YIELD GIVEN NUMBERS OF B.TH.U. PER PENNY OF FUEL COST

Cost of Coal Delivered. Shillings.	Permissible Ash Content of Coal per cent. Yield per penny =		
	100,000 B.Th.U.	120,000 B.Th.U.	140,000 B.Th.U.
5	34·8	32·8	31·2
6	33·4	31·3	29·3
7	32·2	29·8	27·4
8	30·9	28·1	25·4
9	29·4	26·6	23·4
10	28·1	24·7	21·3
11	26·7	23·0	19·1
12	25·1	21·0	16·7
13	23·7	19·1	14·4
14	22·3	17·0	11·5
15	20·4	15·0	9·1
16	18·7	13·4	6·3
17	17·0	10·4	3·2
18	15·3	8·2	—*
19	13·5	5·5	—*
20	11·3	2·9	—*

* Impracticable.

cost, the permissible ash contents of the coal, at different delivered costs, may be read from the chart and are given in Table 165.

From the figures in Table 165 it would appear that the value of a coal for steam generation does not begin to decrease rapidly until the ash content of the coal exceeds about 13 per cent.

This is further illustrated by the results of boiler trials (*Fuel*, 1927, 6, 563) in which an average efficiency of 82·1 per cent. was obtained with three cleaned slacks (ash = 5·3, moisture = 11·9 per cent. average) and 82·6 per cent. with three unwashed slacks (ash = 12·5, moisture = 7·0 per cent. average). The conditions were similar, but the average size of the coal was slightly more favourable to the unwashed slacks.

We have no figures to assess the values of coals of different ash content when they are to be used for pulverised-fuel firing. The relative value of coals is then much nearer to their net calorific values. The presence of incombustible matter in the coal leads to difficulties of operation rather than to out-of-pocket expenses, and it is impossible, at present, to convert into an item of cost the potential nuisance caused by the emission of small particles into the atmosphere, or the effect on heat transmission of a coating of mineral matter of unknown composition (whilst hot) on the water tubes.

Because at present it is possible, in certain cases, to discharge the bulk of the ash as dust with the flue gases, the costs of ash disposal are uncertain and depend largely upon the method of removal of the ash. The principal expenses in using coal high in ash content are the cost of pulverisation and the cost of more frequent renewal of the refractory lining of the boiler. The cost of pulverisation varies according to the moisture content and nature of the coal used, but a figure of 2*s*. 6*d*. per ton may be regarded as approximately the cost in plants of average size (unit system of firing).

THE VALUE OF CLEAN COKE FOR BLAST-FURNACE OPERATION

The disadvantages of the mineral matter contained by the coke fed to a blast-furnace were summarised by Evans (*Journ.* West of Scot. Iron and Steel Inst., 1924–5, vol. 33) as follows :—

1. It reduces the fixed carbon in the fuel.
2. It necessitates the use of additional fuel and limestone for smelting.
3. It reduces the hardness and resistance to abrasion in the coke produced (especially from inferior coking coals), and thus increases the proportion of breeze.
4. It reduces the available heat in the hearth.

A further serious disadvantage of the mineral matter in coal when fed as coke to the blast-furnace is the liability of the iron to be contaminated by other elements, such as sulphur, phosphorus and arsenic. A decrease in the ash content of the coke :—

1. Decreases the coke consumption.
2. Increases the output and therefore reduces standing and capital charges per ton of pig.
3. Reduces the labour cost.
4. Increases the ease of operation of the furnace.
5. Improves the quality of the iron.

It is not easy to evaluate the additional cost imposed upon blast-furnace operations by the ash in the coke, because of the multiplicity of the factors involved; but certain estimates have been put forward from time to time of the quantitative effects of the ash. These estimates have usually dealt with specific points of the general problem, and made no allowance for the increased capacity and ease of operation with a clean fuel and the capital charges saved thereby.

Apart from its lower thermal value, the principal objection to the ash is the increased amount of slag produced, and the extra quantity of limestone and fuel required to convert the ash into slag. The constituent of the ash to which this may be chiefly attributed is the silica. Assuming that the coke contains 10 per cent. of ash, and that the ash contains 40 per cent. of silica, the extra amount of silica (over and above that contained in the iron ore) charged into the furnace is equivalent to 1 cwt. of silica per ton of pig produced. Taking the figures of Ridsdale (*Journ.* Iron and Steel Inst., 1920, 101, 1, 176), the extra solids required to be handled for a slag basicity of 2 amount to ½ ton for every ton of pig. Ridsdale's figures for the effect of charging 1 cwt. of silica into the furnace are as follows:—

Slag basicity	2·0	1·8	1·7	1·5
Limestone required	4·12	3·68	3·47	3·04 cwt.
Coke required	2·42	2·20	2·09	1·86 ,,
Total extra solids charged . .	6·54	5·88	5·56	4·90 ,,
Slag produced	3·47	3·20	3·09	2·82 ,,
Total solids to handle . . .	10·01	9·08	8·65	7·72 ,,

Lewis (*Journ.* West of Scot. Iron and Steel Inst., 1924–5, 33, 2) calculates the additional cost resulting from the presence of the ash-forming materials in a splint coal used for Scotch blast-furnace practice. Taking a standard limestone of composition:—

CaO	54·32 per cent.
CO_2	42·68 ,,
Impurities	3·00 ,,

the lime available as flux is 51·44 per cent., for 2·88 per cent. is required as flux for the impurities in the limestone itself. Assuming that the coal contains 5 per cent. of ash and that the ash contains 8 per cent. of lime, a portion of the ash is self-fluxing, and with a slag containing 49 per cent. of lime, the amount of self-fluxing ash is—

$$\frac{8}{100} \times 5 \times \frac{100}{49} = 0{\cdot}82 \text{ per cent. of the coal.}$$

The remaining mineral matter, amounting to 4·18 per cent. of the coal, requires

$$\frac{4 \cdot 18 \times 49}{(100 - 49)} = 4 \cdot 01 \text{ per cent. of lime,}$$

$$\text{or } \frac{4 \cdot 18 \times 49}{100 - 49} \times \frac{100}{51 \cdot 44} = 7 \cdot 81 \text{ per cent. of limestone.}$$

From this calculation it follows that 7·81 cwt. of limestone are required to flux the 1 cwt. of ash in the coal (5 per cent.), and that the slag made from the coal ash alone amounts to 0·82 + 4·18 + 4·01 = 9·01 per cent. (nearly 2 cwt. per ton of coal). Taking the estimate of Joseph and Read (*Trans.* Amer. Iron and Steel Inst., May, 1924) that every pound of slag requires over half a pound of coke to melt it, it follows that the slag from the coal ash requires $9 \cdot 01 \times \frac{1}{2} = 4 \cdot 5$ per cent. of the coal for fusion purposes.

The loss of carbon is not confined to the amount required to flux the coal ash, but a further quantity is required to flux the impurities in the extra limestone that must be added, and the solution of carbon in the carbon dioxide from the limestone causes an additional loss. Taking these losses into account, Lewis concludes that, for every 1 per cent. of ash in the coal above 5 per cent., the value of the coal is reduced by 9·4*d*. per ton. Assuming this figure, and allowing for other extra expenses, the extra cost per ton of pig iron made is equivalent to an additional cost per ton of pig, as follows :—

Per cent. Ash in Coal.	Extra Cost, pence.
5	—
6	16·83
7	34·44
8	52·90
9	72·28
10	92·62
11	114·02
12	136·56

A number of similar calculations have been made. For example, Thau (*Stahl und Eisen*, 1922, 42, 1155 and 1242) states that the amounts of extra limestone required per ton of pig with varying ash contents of the coke are as follows :—

Coke. Ash Content per cent.	Coke required, kg.	Limestone required, kg.
5·2	1,134	118
8·5	1,200	176
9·3	1,219	192
10·0	1,235	205

He also gives the operating cost with these four cokes, and the extra cost of materials, limestone being valued at 300 mk. per ton and coke at 1,350 mk. per ton. To convert the figures to English currency, an exchange rate of 1,200 mk. to the £ is assumed, making the cost of coke *22s. 6d.* per ton, and of limestone *5s.* per ton.

The savings effected by using coke with 5·2 per cent. of ash, instead of coke with 8·5, 9·3 or 10·0 per cent., are as follows :—

Coke. Ash Content per cent.	Saving in Cost (pence).			Total Saving per ton of pig.
	Coke.	Limestone.	Operation.	
8·5	17·8	3·5	7·9	29·2
9·3	22·9	4·4	10·2	37·6
10·0	27·2	5·3	12·1	44·6

The saving is therefore about *9d.* for every 1 per cent. reduction in ash content.

Calculations by French and Belgian authorities have been made recently by Deladrière (*Rev. Univ. des Mines*, 1921, 9, 93), Derclaye (*Rev. Univ. des Mines*, 1923, 18, 345), and Lilot (*Rev. Univ. des Mines*, 1926, 10, 39).

Deladrière took a slag basicity of 1·7 and calculated that, for every kilogram of dry coke used, the heat lost in converting the ash in the coke into molten slag was $450 + 1237\ (1{\cdot}7s - c)$ cals. per 1 per cent. ash, *s* and *c* being respectively the percentages of silica and lime in the ash of the coke. His calculation took into account the added amount of limestone used and of slag produced, the heat required to decompose the limestone, and the sensible heat of the carbon dioxide formed.

If 25 cwt. of coke are required per ton of pig produced, and the coke ash contains 50 per cent. of silica and 6 per cent. of lime, the heat loss for every 1 per cent. of ash would be

$$\left\{450 + 1237\ (1{\cdot}7 \times 50 - 6)\right\}\frac{25}{20} \times 1{,}000$$

$$= 122 \times 10^6 \text{ cals. per ton of pig produced,}$$

taking 1,000 kg. as being equivalent approximately to 1 ton. This quantity is equal to the heat evolved by the combustion of about 0·35 cwt. of coke.

Derclaye described a series of tests with cokes made from unwashed coal and from coal washed to half its original ash content (*demi-lavé*). The *demi-lavé* coke contained 10·5 per cent. of ash. Derclaye's tests, extending over a period of five years, compare the

effects of using coke containing 10·5 per cent. of ash with other qualities of coke, chiefly a mixture of clean coke containing 10·5 per cent. of ash and dirty coke containing 20 per cent. Some of the average results obtained over four years' working are given in Table 166.

TABLE 166.—BLAST-FURNACE OPERATION WITH COKE MADE FROM WASHED AND UNWASHED COALS

	1st Year. ⅓ Washed ⅔ Unwashed.	2nd Year. ⅚ Washed ⅙ Unwashed.	3rd Year. All Washed.	4th Year. All Washed.
Output per day (tons)	131·5	148·7	148·9	163·3
Coke used per ton pig (kg.)	1,205·0	1,075·5	1,034·0	968·6
Total yield (per cent.)	28·5	28·3	31·3	32·1
Temp. of blast (degree C.)	690	847	863	882
Pressure of blast (cm. mercury)	21	18	16·5	16·5
Temp. at furnace top (degree C.)	89	76	63	54
Ratio CO_2 : CO	0·72	0·75	0·76	0·76
Analysis of iron: Si (per cent.)	0·64	0·54	0·44	0·42
Mn. ,,	1·20	1·30	1·21	1·22
P ,,	1·90	1·88	1·85	1·87
S ,,	0·07	0·06	0·06	0·05
Analysis of slag: SiO_2 ,,	29·17	28·85	28·86	28·80
Al_2O_3 ,,	18·28	18·10	18·67	18·60
CaO ,,	42·91	43·10	43·25	43·20
Basicity	1·59	1·58	1·60	1·59

In addition to an increased output, the coke consumption per ton of pig was reduced, the blast pressure was decreased, and the quality of iron was improved. The net gain in using coke made from washed coal, allowing for the cost of materials, cost of operation, wages, etc., was found to result in a decreased cost per ton of pig of 2·09 fr. (pre-war rate of exchange), equivalent to 1*s.* 8*d.* per ton of pig. The improved quality of the pig was considered to be worth 1·80 fr., or 1*s.* 5*d.* per ton more than previously. The net gain is therefore 3*s.* 1*d.* for an ash reduction in the coke of 9·5 per cent.

Lilot, taking account of the weight of slag produced, the loss of calorific value, the loss of manganese in the slag and the decreased output, calculates the loss of heating value resulting from the ash in blast-furnace coke, and concludes that if a coke with an ash content of 9 per cent. is worth 130 fr. per ton, a coke containing 13 per cent. of ash is worth only 116¾ fr., a decrease of approximately 2*s.* in the price for an increase of 4 per cent. in the ash content.

Evans (*loc. cit.*) describes two experiments made in British practice to determine the effect on output and fuel consumption of

decreased ash in the coke, the furnace being operated so that (*a*) the silicon content of the iron was unimportant and (*b*) the iron made was of the highest possible quality. The coke normally used contained approximately 12·5 per cent. of ash, 0·7 per cent. of sulphur and 82 to 84 per cent. of "fixed carbon," and the coke consumption per ton of hæmatite iron was 24 cwt., with a burden of ore and limestone of 46 cwt.

With a burden of 45 cwt., and coke containing 8·5 per cent. of ash, the furnace immediately began to drive faster, and to obtain the maximum output, the silicon content of the iron was allowed to fall from 2 to 0·93 per cent. The output was increased 30 per cent. and the coke consumption fell from 24 to 19·4 cwt. per ton.

In the second test, the silicon content of the iron was maintained, and it was found to be easy to maintain 3 per cent. of silicon in the iron. The output of the furnace was simultaneously increased by 12 per cent. (and later by 20 per cent.) with a drop in coke consumption of 0·7 cwt. per ton of pig.

Evans, in a private communication, calculates the additional quantity of coke required because of an increase of 1 per cent. in its mineral matter content as 36·2 lb. per ton of pig. He assumes an ash content in the coke of 10 per cent., the ash containing 40 per cent. of silica, and a coke consumption of 2,400 lb. per ton of pig. The silica in the ash then amounts to charging into the furnace approximately 100 lb. of silica. With a slag basicity of 1·5, each additional 1 per cent. of ash means 10 lb. of additional silica, or 28 lb. of extra slag. Assuming that 1 lb. of slag requires 0·227 lb. of carbon to melt it, the additional carbon required is $28 \times 0{\cdot}0227 = 6{\cdot}4$ lb.

The coke, however, contains only 85 per cent. of carbon, or 2,040 lb. of carbon per ton. The total carbon required is therefore 2,046·4 lb. The carbon content of the coke is, however, reduced from 85 to 84 per cent. because of the additional 1 per cent. of ash, and the total amount of coke required becomes

$$\frac{2{,}046{\cdot}4}{0{\cdot}84} \times 100 = 2436{\cdot}2 \text{ lb.}$$

For chemical reasons, therefore, an addition of 1 per cent. to the ash content of the coke increases the coke consumption by a minimum of 36 lb. per ton of iron. Evans further states: "To this would have to be added the physical effects and, in practice, a reduction of 4 per cent. of ash in the coke has been found to increase output and reduce fuel consumption far more than would be necessitated simply from the point of view of this calculation."

In the same communication Evans considers empirically the effect of sulphur in coke. For every 0·1 per cent. of additional sulphur in the coke, 2·4 lb. of sulphur are charged and 8·4 lb. of limestone is required to neutralise it. The 4·7 lb. of slag produced require only 1·2 lb. of coke for smelting.

On the other hand, if it be assumed that the Cleveland slag con-

tent of 1·5 per cent. of sulphur is a measure of the solubility of calcium sulphide in the slag, it follows that the amount of slag made must be increased when additional sulphur is charged, in order to maintain the same sulphur content of the slag.

Normally, making 30 cwt. of slag per ton of pig, with a sulphur content of 1·5, the sulphur in the slag weighs $1{\cdot}5 \times 30 \times 112 =$ 50·4 lb. The additional 2·4 lb. of sulphur makes a total of 52·8 lb., which requires

$$\frac{52{\cdot}8}{1{\cdot}5} \times 100 = 3{,}520 \text{ lb.}$$

of slag, an increase of 160 lb., necessitating an extra charge of 40 lb. of coke.

Evans suggests that the actual increased coke consumption because of sulphur in the coke, is probably mid-way between the amounts suggested by these two calculations. When the blast-furnace manager is dissatisfied with the slag, he increases the amount of limestone to restore the normal fracture. By so doing, the basicity of the slag is increased, and the slag therefore has a higher solubility for sulphur. By increasing the basicity, however, the volume of slag is increased and its melting-point is elevated, so that there is an increased fuel consumption. Increasing the basicity from 1·5 to 1·6 requires the addition of 180 lb. of limestone, which, for its fusion, requires 29 additional pounds of coke.

The effect of varying ash in the coke has been investigated recently by the Consett Iron Co., Ltd., and the conclusions were published by Gill (*Journ.* Iron and Steel Inst., 1927). Gill found that the greatest trouble arose from fluctuations in the average ash content of the coke supplied to the furnaces. He also gives the results of two series of tests, made independently and at different times, in which coke of uniform ash content was supplied to the furnace for periods of five weeks. The results are given in Table 167.

TABLE 167.—EFFECT OF VARYING ASH IN COKE

	Test 1.	
	Period 1.	Period 2.
Average ash in coke (per cent.)	12·15	9·76
Average weekly make of pig (tons)	3,311	3,819
Tonnage of No. 3 grade (tons)	210	134
Percentage of No. 3 grade (per cent.)	6·34	3·51
Average pressure of blast (lb. per sq. in.)	5¾	4¾
Percentage increase in make (per cent.)	—	15·4
Reduction in coke consumption (cwt. per ton pig)	—	1·0

	Test 2.	
	Period 1.	Period 2.
Percentage ash in coke (per cent.)	12·5	8·7
Average weekly make of pig (tons)	950	1,190
Percentage increase in make (per cent.)	—	25·25
Reduction in coke consumption (cwt. per ton pig)	—	1·76

Averaging the results of these two tests, it is found that, for a 1 per cent. reduction in the ash content of the coke, the output of the furnace is increased by 6·4 per cent. in the first test and 6·7 per cent. in the second, whilst the coke consumption was reduced by 0·42 and 0·47 cwt. per ton of pig respectively. Gill estimated that the saving in coke alone amounted to 1*s*. 3*d*. per ton of pig for a 3 per cent. reduction in the ash content of the coke. Prof. H. Louis, in the discussion on Gill's paper, stated that if other factors were taken into account the saving would amount to at least 2*s*. 3*d*. per ton of pig.

Although there is some variation between the estimates of the authorities quoted, there is a fair agreement between the estimated reduction of the amounts of coke required with different percentages of ash.

Authority.	Extra Coke required for each Increase of the Ash Content by 1 per cent. Cwt. per Ton of Pig.	Range of Ash Contents per cent.
Thau	0·40	5–10
Deladrière	0·35	5–12
Gill	0·45	8–12
Evans (*a*)	0·18	8·5–12·5
(*b*)	1·05	8·5–12·5
(*c*)	0·35	10–11

The three figures of Evans are : (*a*) When the quality of iron was improved, (*b*) when no attention was paid to the silicon content of the iron, and (*c*) calculated.

If coke with 10 per cent. of ash has a value of 20*s*. per ton, the value of coke with only 3 per cent. of ash would, on these estimates, be worth over 3*s*. more per ton, irrespective of its large additional worth in increasing the furnace output, decreasing the amount of limestone to be added and of slag to be removed, and generally

reducing the operating costs, whilst producing an iron of higher quality. It is quite possible that in actual operation a coke with an ash content of only 3 per cent. might be worth 7*s*. or 8*s*. more per ton than coke containing 10 or more per cent. of ash.

THE COMMERCIAL VALUE OF COAL. PURCHASE ON CALORIFIC VALUE

The statement that coal has a higher value to the consumer, the lower its ash content, may be accepted without reservation, but low ash content is not the only quality of a high-grade fuel. The principal factor governing the commercial value of a coal is its suitability for the consumer's purpose. A coal which is an excellent coal for steam-raising in a Scotch boiler would have scarcely any value for the manufacture of blast-furnace coke, and a good coking coal is almost useless for the manufacture of water gas. But provided that a coal is suitable for a given purpose, it will fulfil that purpose more satisfactorily the lower its ash content.

Apart from considerations of size, the principal factors which govern the suitability of a coal for any given purpose are its volatile matter content, its coking properties and the fusibility of its ash. These properties depend upon the chemical composition of the coal and of its inorganic constituents. In order to correlate the chemical composition of different coals with their observed behaviour on combustion or on submission to thermal decomposition (carbonisation or gasification), various schemes for the classification of coals have been proposed from time to time. Those which are now regarded as having the greatest value either for scientific purposes or in connection with the industrial utilisation of the coal are Seyler's classification and Parr's classification.

Seyler (*Proc.* S. Wales Inst. of Eng., 1900, 21, 483 ; *ibid.*, 1901, 22, 112 ; *Fuel*, 1924, 3, 15, etc.) divides coals into genera according to their hydrogen contents, and each genus is divided into different species according to the carbon contents of the coals. The classification thus provided enables the properties of the coals to be predicted with remarkable success from their ultimate analyses.

Parr's classification is based on the calorific values of the coals and their volatile matter contents, and the groups into which coals are found to fall when classified on this basis are found to correspond, roughly, with the groups in Seyler's classification. Parr's classification followed upon the observation by Lord and Haas (*Trans.* Amer. Inst. Min. Eng., 1898, 27, 259) that coals from different parts of certain fields had the same calorific value when referred to a "unit-coal" basis, results confirmed by numerous other workers in America, *e.g.*, Noyes (*Journ.* Amer. Chem. Soc., 1898, 20, 285), and in England by Drakeley (*Trans.* Inst. Min. Eng., 1918, 56, 45), who has shown that the calorific value of certain British coals may be expressed

$$C.V. = 147{\cdot}6\left(100 - \frac{100A}{89}\right)$$

where A is the ash content on the dry basis, and by Whitaker (*Trans. Inst. Min. Eng.*, 1924, 67, 199), who has found that a similar formula

$$C.V. = 145\left(100 - \frac{100A}{87}\right)$$

is generally applicable to certain Nottinghamshire and Derbyshire coals.

All schemes for the classification of coals depend upon the figures provided by the analysis of coals. In order that one coal may be compared with another, it is necessary to refer the results of analyses of each coal to some uniform fundamental standard, for no coal occurs in nature as a "pure" substance, uncontaminated with other matter than the actual substance of the coal.

For the purpose of the scientific classification of coals it is necessary to eliminate the effects of those ingredients of pit coal which are essentially extraneous to the pure coal substance, and for this purpose several formulæ have been proposed. The most obvious ingredients whose effect must be eliminated are the ash and free moisture of the coal. But there are also other constituents of coal whose effect is undesirable if the coal is to be classified on a fundamental basis, such, for example, as the sulphur present as pyrites and the carbon present as carbonate. Each of these ingredients either increases or lowers the observed calorific value of unit mass of the "pure" coal substance, or the carbon content determined by combustion. Each of these substances, moreover, undergoes a change in weight when the coal is burned, so that its effect cannot be eliminated simply by reference to the ash content of the coal. Similarly, the determination of the ash content of the coal makes no allowance for the "water of constitution" of the shale or hydrated salt that may be present in coal, the water being driven off on incineration. As yet there is no accepted standard whereby proper allowance can be made for these substances associated with the "pure" coal, and scientific classification is hampered by the lack of such an accepted standard.

Standards have been proposed by Seyler and by Parr as a means of expressing analytical results in such a form that they may be utilised for scientific classification. Seyler suggests that the results should be corrected to "pure coal" by multiplication by the factor

$$\frac{100}{100 - (A + W + S)}$$

where A, W and S are respectively the ash, moisture and sulphur contents of the crude coal. Parr (*Bull.* 37, Univ. of Ill. Eng. Expt. Stn., 1909), to calculate the calorific value of the coal proposes a formula

$$\text{C.V.} = \frac{\text{B.Th.U. indicated} - 5{,}000\ \text{S}}{1{\cdot}00 - \left(\text{M} + 1{\cdot}08\ \text{A} + \frac{22}{40}\text{S}\right)}$$

M, A and S, being the weights of moisture, ash and sulphur in unit mass of the coal. Parr thus makes allowance in his calculation for the heating value of the sulphur and for the presence of hydrate water in the ash (A being multiplied by 1·08 for this purpose).

These formulæ, or a compromise between them, or some other formula of the same type, are required as a basis for the purpose of determining the class or group into which the coals fall, and in order to indicate their suitability for a certain industrial use. But such formulæ do not enable an estimate to be formed directly of the relative merits of two coals which, by classification, are both known to be of the same industrial class. Thus, if two coals fall into the true steam coal class, it is desirable to distinguish their relative values for steam-raising purposes.

In order to draw this distinction, it has frequently been suggested that a determination of the lower calorific value of the coal is sufficient, and, in general, this is correct. Because they fall into the same class, the two coals will be composed more or less of the same " unit " coal, they will each contain approximately the same amount of volatile matter, each will have roughly the same ratio of carbon content to hydrogen content, and they will behave similarly on combustion or on thermal decomposition. The calorific value is therefore a guide to their potential usefulness, especially if they are required solely as sources of heat, for steam-raising purposes, for example.

But supposing that the two coals do not fall into the same group in any system of classification, they may then be widely dissimilar in many properties. One may be an anthracite and the other a sub-bituminous coal, and their commercial values for some one especial purpose will be widely different. Yet it is possible that, on account of differences in their ash contents, they will have the same indicated calorific value. Consequently, without some reliable indication of the position that a coal will occupy in, say, Seyler's or Parr's classification, a mere statement of the calorific value of the coal may be valueless as an index of its commercial usefulness. The sale of coal by calorific value alone should therefore not be accepted as an infallible guide to its value. In order to establish its utility, it is necessary to state not only the calorific value, but also its ash content, its moisture content and its volatile matter content.

Moreover, though two coals may fall into the same group of a classification and may have identical calorific values, it is not certain that one will be of the same value as the other, because one may have a very fusible ash, the other an infusible ash ; one may be a slack with a fairly uniform distribution according to size, and the other may contain a preponderance of fines.

At present, therefore, whilst the sale of coal on the basis of calorific value is to be welcomed as a step in the direction of the more scientific utilisation of fuel, it cannot be regarded as fulfilling entirely the demand for a numerical formula or basis to assess the absolute value of a coal. Moreover, useful as the calorific value may be as an indication of the number of available B.Th.U. per lb. of coal, it must be remembered that the determination of calorific values in the laboratory is not an easy determination to make and quite large errors may be introduced by reason of the human element.

As a rule, it is sufficient to state the ash and moisture contents of a coal. Provided that two coals fall into the same group of a classification, their ash and moisture contents are as good a guide to their relative merits as are their calorific values, and unless they fall into the same group of classification, the mere statement of their calorific values is of little importance for comparison.

CHAPTER XXXIII

THE ECONOMICS OF COAL CLEANING

It is doubtful whether there is a single industrial process dependent upon the combustion of coal which cannot be carried out with a clean coal more efficiently and at a lower operating cost than with a coal containing larger quantities of incombustible matter. Some of the advantages to be derived from clean coal have been considered in Chapter XXXII., and the benefits of clean coal to the consumer make it desirable to supply him with the purest possible quality. It was, indeed, to lessen the consumer's difficulties that coal cleaning was first introduced.

The producer, however, has a natural wish to sell the maximum amount of the material won at the pit face and brought to the surface as coal. It is seldom possible for the miner to work a seam without including in his winnings a quantity of dirt, some of which is loaded and sent to the surface, though some is sorted out from the coal underground and left in the goaf. In leaving the dirt in the pit, the producer may sustain a loss of coal, for it is impossible for the miner in the dim light of a miner's lamp accurately to distinguish between coal and dirt.

It is usually considered that, in Great Britain, over 90 per cent. of the possible extraction of coal is made,* and of the remainder little of the loss is avoidable, as it is mainly due to leaving unworkable pillars. Most of the loss which may be considered avoidable occurs in South Wales, where the small coal from certain seams is screened out by forking and returned to the goaf. It has been suggested that these smalls should be brought to bank and cleaned, but Hay ("Britain's Fuel Problems," London, 1927) calculates that this practice would mean a loss of at least seven shillings per ton of coal so treated.

The removal of dirt from the raw coal brought to the surface (for which the costs of winning, loading, haulage and winding have already been incurred) involves the colliery company in the cost of picking and washing and in other incidental expenses. Moreover, during these processes, the saleable coal tends to be reduced in size and some of the output is rejected as refuse. The actual cost of production per ton of the marketable coal is increased in proportion as the amount of refuse removed from it is increased, and the market

* In the U.S.A. an average extraction of only 65 per cent. is made, and of the residual material 19 per cent. is recoverable ("U.S. Coal Commission Report," Vol. III, p. 1841).

value of the coal is lowered because, at present, large coal commands a higher price than small coal.

It is therefore inevitable that the producer should consider the optimum recovery of saleable coal of more importance than the marketing of the cleanest possible product. When relatively little coal is sold to a specification, it is in his interest not to remove from the coal such dirt as the buyer is willing to purchase. If the consumer is satisfied with coal containing 8 per cent. of ash, and will not pay more for a purer quality of coal, it is against the colliery company's interest to supply him with coal containing 4 per cent. of ash (except that his customer is less likely to pay the same price for coal from another source). It is reasonable, therefore, that the colliery proprietor should look upon coal cleaning with disfavour except in so far as it enables him to obtain or retain markets which he would otherwise lose, and that, unless the consumer is prepared adequately to remunerate him, he should confine his washing operations to that minimum which permits of the sale of his coal.

On the other hand, the consumer should be prepared to recognise that coal of low ash content is a more useful commodity than a coal of higher ash content. He should further recognise that a very clean coal has a higher heat value and that its potential heat can be realised at a lower cost. This he does recognise, but only to a limited extent. He will, for example, buy the one of two coals which has the lower ash content of the two, provided that it will equally well suit his purpose and that the price is no higher. But few consumers will pay a shilling more per ton of coal because the coal contains, say, 3 per cent. less ash, although for many purposes a reduction of 3 per cent. in the ash content would increase the value by more than one shilling. It is not suggested that the consumer pays no attention to the relation between the ash content and the price of his coal, but in only rare cases does he realise to what extent it is to his advantage to be supplied with a low ash coal.

The cost to the producer of cleaning coal to a certain ash content is the sum of a number of contributory expenses and losses. In the first place, the cost of passing the raw coal through the washery or cleaning process and the cost of maintaining the plant in running order must be met. To this must be added the interest on capital, depreciation, rates and insurance, and the cost of supervision (which includes a proportion of the salaries of the general manager of the colliery, the chief engineer and the chemist). The colliery chemist is usually required to undertake control testing, and the cost of materials for this should also be debited against the washery.

Another, and important, consideration in assessing the cost of coal cleaning, is the possibility of troubles in the washery which may lead to irregular working and especially to periodical stoppages. A serious stoppage may, in certain circumstances, lead to the complete stoppage of winding, for if the washery is out of action and there is insufficient storage accommodation in the sidings, it may be useless

to bring coal to the surface which, without cleaning, has a reduced value. In this event, the sum to be debited against the washery is a serious item in the weekly or annual cost.

By far the most serious cost, however, is the loss of some of the pit's output in the form of refuse, and the loss of combustible matter in the refuse. With the most efficient washing, the refuse removed does not contain 100 per cent. of ash. The shale associated with the coal seldom contains more than about 80 per cent. of ash, with a further content of 10 per cent. of combined water. Moreover, with practically all modern processes, it is impossible to prevent the passage of a certain quantity of useful coal into the refuse. The purest coal that can be prepared by washing is never entirely free from ash ; it always contains fixed ash by reason of the mineral matter more or less intimately mingled with the coal substance. Although coals are known in which the fixed ash in the coal substance may be as low as 0·7 per cent., usually the figure is 1 to 3 per cent. for the bright portions of coal and somewhat higher for the dull hard portions.

If the raw coal consisted of particles, some yielding no ash on incineration and others yielding 100 per cent. of ash, by the most efficient methods of cleaning, the ash content of the raw coal, x per cent., could be reduced to y per cent. in the washed coal by removing $\frac{100\,(x-y)}{100-y}$ per cent. of its gross weight. But if the raw coal is made up of particles of coal each containing a per cent. of ash and particles of shale each containing b per cent. of ash, the minimum loss of output is $\frac{100\,(x-y)}{b-y}$ per cent.

It is well known, of course, that pit-coal does not consist simply of a number of individual particles with one given ash content and a number of other particles with another given ash content, but that it contains particles of more or less pure coal of specific gravity less than 1·35 and more or less pure dirt of specific gravity greater than 1·6. Between these two specific gravities is a proportion of middlings. This does not, however, invalidate the calculation, because, in commercial coal cleaning, the control of the process is based (and the efficiency of operation is often calculated) upon the results of float-and-sink analyses. The operator knows that, if all the particles below a certain specific gravity pass into the clean coal, he will obtain a satisfactory product, and that if all the particles of higher specific gravities pass into the refuse he is sustaining no loss of saleable or useful output. Just as it may be satisfactory to base the control of the washery on the results of float-and-sink tests at some one or more arbitrary specific gravity, so too, for the purpose of our calculation, it is permissible to regard the clean coal as a mass of particles (the floats) with a certain mean ash content. The refuse

may be regarded as another mass of different particles (the sinks), also with a mean ash content.

In industrial practice, it is never possible to effect a sharp and complete separation between the particles comprising the floats and those comprising the sinks, and it is usual to consider that the washing is performed with a satisfactory over-all efficiency if the clean coal contains not more than 2 per cent. of sinks and the refuse contains not more than 2 per cent. of floats.

A better way of expressing the efficiency of washing is by the ash contents of the washed coal and the refuse. Thus, if a float-and-sink test indicates that the coal can be divided into two fractions, one including A particles with a mean ash content of a per cent., and the other, B particles with a mean ash content of b per cent., it may be decided so to wash the coal that, as far as possible, the A particles with a per cent. of ash are collected as clean coal, and the remaining B particles are rejected. The limitations imposed by inaccuracies of operation on the commercial scale, however, prevent the maximum efficiency from being obtained, and, under average conditions, the washing is as nearly perfect as circumstances permit if the fractions obtained by washing contain : clean coal, $a + 0{\cdot}5$ per cent. of ash ; and refuse, $b - 3$ per cent. of ash. Thus the clean coal may be expected to contain 0·5 per cent. of ash more than the theoretical minimum, and the refuse 3 per cent. of ash less than the theoretical maximum.

The loss of output sustained is then

$$\frac{100\,(x - y - 0{\cdot}5)}{b - y - 3{\cdot}5},$$

or, in the case we have cited, where $y = a$, the minimum practicable loss is

$$\frac{100\,(x - a - 0{\cdot}5)}{b - a - 3{\cdot}5},$$

x being, as before, the ash content of the raw coal.

This may be illustrated by an example. The float-and-sink analysis of a Welsh coal is given in Table 168.

TABLE 168.—FLOAT-AND-SINK ANALYSIS. WELSH COAL

S.G.	Per cent. of Total.	Per cent. Ash in Fraction.	Cumulative Ash Content of Float per cent.	Ash Content of Total Sink per cent.
<1·35	64·3	3·8	3·8	41·0
1·35–1·4	8·6	9·8	4·5	50·9
1·4 –1·5	6·0	15·7	5·3	60·9
1·5 –1·6	3·2	37·5	6·6	65·1
>1·6	17·9	65·1	17·0	—

If it were desired to collect all the coal lighter than a specific gravity of 1·35, the optimum products would be 64·3 per cent., of clean coal containing 3·8 per cent. of ash and 35·7 per cent. of refuse containing 41·0 per cent. of ash. In practice, the washery could be regarded as performing its duty with a reasonable efficiency if the clean coal contained 4·3 per cent. of ash and the refuse 38·0 per cent. In these circumstances, the loss of output would be

$$\frac{100\ (17{\cdot}0 - 4{\cdot}3)}{41{\cdot}0 - 3{\cdot}8 - 3{\cdot}5} = 37{\cdot}7 \text{ per cent.}$$

as compared with a theoretical amount of 41·0.

Similarly, if it were desired to recover all the coal of specific gravity less than 1·5, the products, with satisfactory operation, would be: Clean coal containing $5{\cdot}3 + 0{\cdot}5 = 5{\cdot}8$ per cent. of ash, and refuse containing $60{\cdot}9 - 3{\cdot}0 = 57{\cdot}9$ per cent. of ash, and the loss of output would be

$$\frac{100\ (17{\cdot}0 - 5{\cdot}8)}{60{\cdot}9 - 5{\cdot}3 - 3{\cdot}5} = 21{\cdot}5$$

against a theoretical minimum loss of 21·1.

This formula for minimum output loss with a reasonable margin for inaccuracies of operation, enables a calculation to be made of the minimum amount by which the consumer must reimburse the producer if he wishes to be supplied with a cleaner coal. If the pithead price of the coal is P shillings per ton, and the cost of putting 1 ton of coal through the washery is Q shillings, the cost to the producer per ton of washed coal is

$$\frac{(P + Q)\ (b - a - 3{\cdot}5)}{b - x - 3} \text{ shillings,}$$

or, expressed in words,

$$\left\{(\text{Pithead cost of coal}) + (\text{cost of washing})\right\}$$

$$\left\{\frac{(\text{Ash content of sinks}) - (\text{ash content of floats} - 3{\cdot}5)}{(\text{Ash content of sinks}) - (\text{ash content of raw coal}) - 3}\right\}$$

This value will vary with the nature of the coal and with the extent to which the ash content of the coal is reduced. The value given by this formula is, however, a useful basis upon which the producer could negotiate with the consumer, the producer offering the consumer a cleaner coal if he is prepared to offer him a correspondingly enhanced price.

The use of the formula necessitates a knowledge of the float-and-sink analysis of the coal. Considering a coal with the analysis given in Table 168, and taking the pithead cost of the coal (excluding washing) as £js. 6d. per ton and the overall cost of washing as 6d. per ton of throughput, making the cost of the products delivered from

the washery 10s. per ton, the producer could offer the consumer any of a variety of qualities of coal at given prices, the proceeds of the sale of the clean coal amounting to 10s. for every ton put into the washery. The qualities and prices are given in Table 169.

TABLE 169.—COST OF PRODUCTION OF DIFFERENT QUALITIES OF CLEAN COAL

Ash Content of Washed Coal per cent.	Loss of Output per cent.	Cost of Production *s.* *d.*
4·3	37·7	16 0
5·0	28·0	13 10
5·8	21·5	12 8
7·1	18·3	12 3

The calculation is as follows: For separation at S.G. 1·35, $a = 3{\cdot}8$ (Table 168, col. 4), and the washed coal may be expected to contain $3{\cdot}8 + 0{\cdot}5 = 4{\cdot}3$ per cent. of ash. In these circumstances, $b = 41{\cdot}0$ (Table 168, col. 5), and $x = 17{\cdot}0$ (the ash content of the raw coal). The cost of the washed coal is therefore :

$$\frac{10\,(41{\cdot}0 - 3{\cdot}8 - 3{\cdot}5)}{41{\cdot}0 - 17{\cdot}0 - 3} = 16{\cdot}0 \text{ shillings.}$$

Similarly, for separation at S.G. 1·4, $a = 4{\cdot}5$, $b = 50{\cdot}9$, and $x = 17{\cdot}0$, the washed coal containing $4{\cdot}5 + 0{\cdot}5 = 5{\cdot}0$ per cent. of ash.

The increased cost for the purest qualities of this washed coal is high, because it contains nearly 20 per cent. of " pure " dirt and has a high proportion of middlings. It is also, for the latter reason, fairly difficult to wash, but the majority of well-recognised modern washeries could produce any of the stated qualities of washed coal from it by adjusting the settings of the washer. The best washers are all designed so that a variety of products can be prepared according to the demands of the market.

With a coal containing a lower proportion of dirt and of middlings, the cost of producing washed coal is not so widely different from the cost of the coal delivered to the washer. The difference may be shown by considering the Lancashire coal with the float and sink analysis given in Table 170.

This is a very good coal, containing scarcely any middlings particles and therefore very easy to wash. It should not be difficult to attain an efficiency greater than the standard suggested for average good washing practice. Except for some very special purpose, the coal would only be washed to make a cut at a specific gravity of 1·6, but applying the formula for loss of output and cost of

Maintenance	1·074*d*.
Operation	1·433*d*.

making a total of 2·51*d*. exclusive of interest on capital, management, etc.

Appleyard (*Trans.* Inst. Min. Eng., 1927, 73, 404) gives the costs of operation of the Wardley dry-cleaning plant, in which 6 S.J. pneumatic separators are used to clean 125 tons per hour of coal from 2 in. to 0, the coal being sized into fractions: 2 to 1 in., 1 to $\frac{1}{2}$ in., $\frac{1}{2}$ to $\frac{1}{4}$ in., $\frac{1}{4}$ to $\frac{1}{8}$ in., $\frac{1}{8}$ to $\frac{1}{16}$ in. The coal below $\frac{1}{16}$ in. is by-passed without treatment. Taking an average over five months' operation, the operating and maintenance costs total 2·93*d*. per ton of raw coal.

The cost of operating a South Yorkshire Baum washery of 75 tons per hour capacity, with one wash-box and no provision for rewashing the small coal, has been found to be as follows:—

	Pence per ton of Raw Coal.
Wages. Operation	3·09
,, Maintenance	1·03
Renewals and stores	1·53
	5·65

On a 100 ton per hour plant elsewhere, the figures are stated to be:—

	Pence per ton of Raw Coal.
Wages	1·70
Power and lighting	0·65
Maintenance	0·75
	3·10

For this washer, the cost of renewals is given as 1·0d. per ton.

These figures are exclusive of interest on capital, depreciation and insurance.

The costs for operation and maintenance of these four types of washer may therefore be compared, each having a capacity of 100 to 150 tons per hour.

	Pence per ton of Raw Coal
Blackett	2·51
Rheolaveur	2·68
S.J. pneumatic tables . .	2·93
Baum	3·10

In each case the cost is well under 4*d*. per ton.

By comparison, the capital costs of coal cleaning are relatively high. The capital cost may be expressed in pounds per ton of hourly capacity, and if the cost is £100 per ton per hour (or a total of £12,500 for a plant to treat 125 tons per hour) the capital expenditure is 8*d*. per ton of coal cleaned per annum (assuming a ten-hour washing day and washing on 300 days per annum). Allowing for interest at 10 per cent. and capital redemption in 15 years by a fund carrying 3½ per cent. interest, the capital charge is 1·21*d*. per ton of coal treated.

The capital cost of modern cleaning processes lies, usually, between £100 and £250 per ton of coal washed per hour, so that capital charges may vary from 1·21*d*. to 3·02*d*. per ton treated. Allowing 0·5*d*. per ton washed for insurance, management, rates, taxes, depreciation, etc., the general charges will range from 1·71*d*. to 3·52*d*. per ton of raw coal.

CONCLUSION.

In Chapter XXXII, the question of coal cleaning as it affects the consumer of coal was examined, and in this chapter the subject has been regarded from the producer's aspect. In some respects the convenience of having coal with the minimum amount of ash is recognised by the consumer, but it is not sufficiently well recognised that the advantage is a financial advantage and not only a question of convenience.

The advantages appear to be greatest perhaps in the use of metallurgical coke for blast-furnace operation, but that may be largely due to the fact that the cost of operation of a blast furnace is usually carefully determined and examined, and possible economies suggest themselves more easily than in circumstances where no such careful fuel costing is practised.

To the producer, coal cleaning is a source of expense, both in capital outlay, operating cost and loss of output, but it is frequently a necessary expense in order that the coal may be sold against competitors. At other collieries it is a profitable expense, because it enables the coal to be sold for a considerably higher price.

We have endeavoured to show, with a number of examples, the extent to which the cost of producing clean coal is influenced by the loss of output sustained. The examples (*vide* Tables 168 to 172) showed that the economics of coal cleaning is a subject for separate study at each individual colliery; that it depends considerably more upon the nature of the coal than upon the nature of the washer or upon the cost of washing.

One especial point is brought out by these examples, which is of considerable importance when considered in connection with the present condition of the British export coal trade. Taken by and large, British coals are easier to wash than Continental coals. If,

therefore, a demand for very clean coal could be created on the Continent, the British producer would have an advantage over his Continental competitors. If the costs of bringing the coal to the pithead were, British coal x shillings, and Continental coal $\frac{3}{4}x$ shillings, it is a practical possibility that the production of coal containing, say, 3 or 4 per cent. of ash would, taking into account the loss of output involved in washing, cost the British producer no more, and in very many cases less, than the cost to his Continental competitor. It is no idle statement, therefore, that the salvation of the British export coal trade may lie in the cleaning of coal.

APPENDIX I

THE most important property of a coal from the point of view of coal cleaning is its distribution according to specific gravity and size. We have collected a large number of results of float and sink tests on sized samples of coal from various parts of the world, and a number of these results are given in the following tables. The majority of the results are for British coals, and because of the difficulty of obtaining reliable and representative results from all the coal-producing States of America, we have not included any results for American coals.

Table I.—Float and Sink Results. Sized Samples of British Coals

Coal.	Size (in.).	Per cent. of size	Per cent. by weight and per cent. ash in fractions at different specific gravities.											
			< 1·35 S.G. Weight.	Ash.	1·35—1·40 S.G. Weight.	Ash.	1·40—1·50 S.G. Weight.	Ash.	1·50—1·60 S.G. Weight.	Ash.	> 1·60 S.G. Weight.	Ash.	Total. Weight.	Ash.
DURHAM. Five Quarter	$>\frac{1}{4}$	16·0	74·9	2·7	7·5	10·0	6·0	16·9	2·0	27·4	9·6	68·4	100·0	11·0
	$\frac{1}{4}-\frac{1}{8}$	33·6	73·9	2·5	7·5	10·4	5·6	16·9	1·9	27·3	11·1	67·9	100·0	11·7
	$\frac{1}{8}-\frac{1}{16}$	17·9	69·6	2·3	6·0	10·1	6·2	16·0	2·0	26·4	16·2	69·2	100·0	14·9
	$\frac{1}{16}-\frac{1}{64}$	18·6	65·4	2·0	5·8	9·0	5·0	14·4	2·4	23·9	21·4	69·3	100·0	17·8
	$<\frac{1}{64}$	13·9	—	—	—	—	—	—	—	—	—	—	100·0	18·6
	$>\frac{1}{4}-0$	100·0	70·9	2·4	6·7	9·9	5·7	16·1	2·1	26·2	14·6	68·8	100·0	14·3
"T"	$>\frac{1}{2}$	37·2	77·0	2·7	3·7	9·5	2·3	16·3	2·2	26·0	14·8	63·4	100·0	12·8
	$\frac{1}{2}-\frac{1}{4}$	14·0	81·7	2·4	2·8		1·8		1·2		12·5	67·4	100·0	11·3
	$\frac{1}{4}-\frac{1}{8}$	14·5	76·2	2·3	3·2	9·2	2·0	15·4	1·3	25·5	17·3	70·6	100·0	14·9
	$\frac{1}{8}-\frac{1}{16}$	9·6	70·8	2·2	4·2		2·6		1·8		20·6	70·7	100·0	17·3
	$\frac{1}{16}-\frac{1}{64}$	17·2	66·6	2·1	3·6	9·0	3·6	14·3	1·6	25·6	24·6	68·4	100·0	19·4
	$<\frac{1}{64}$	7·5	—	—	—	—	—	—	—	—	—	—	100·0	24·6
	$>\frac{1}{2}-0$	100·0	74·4	2·4	3·5	9·3	2·5	15·4	1·6	25·8	18·0	68·2	100·0	15·3
"W"	$>\frac{1}{2}$	26·5	76·8	2·6	7·3	12·3	3·5	19·9	3·3	30·8	9·1	58·8	100·0	10·0
	$\frac{1}{2}-\frac{1}{8}$	51·2	75·8	2·9	6·0	11·3	3·8	18·4	2·6	27·3	11·8	63·5	100·0	11·8
	$\frac{1}{8}-\frac{1}{64}$	16·7	70·9	2·2	5·1	9·9	5·8	16·6	2·4	27·2	15·8	64·6	100·0	13·9
	$<\frac{1}{64}$	5·6	—	—	—	—	—	—	—	—	—	—	100·0	17·2
	$>\frac{1}{2}-0$	100·0	74·5	2·6	6·1	11·3	4·4	18·0	2·8	28·4	12·2	62·7	100·0	12·0
Brockwell	$1\frac{1}{2}-\frac{1}{2}$	100·0	—	—	—	—	78·5	6·6	1·1	28·2	20·4	75·2	100·0	20·8

Brockwell	1/2—1/4	8·5	—	—	—	—	90·8	5·0	2·0	28·2	7·2	68·2	100·0	10·0
	1/4—1/8	29·0	—	—	—	—								
	1/8—1/16	23·1	—	—	—	—	93·5	4·1	1·0		5·5	67·2	100·0	7·8
	1/16—1/32	19·4	—	—	—	—								
	1/32—1/64	12·1	—	—	—	—	—	—	—	—	—	—	100·0	9·8
	<1/64	7·9	—	—	—	—								
	1/2—0	100·0	—	—	—	—	92·2	4·5	1·5	28·2	6·3	67·8	100·0	8·8
Busty (1)	1/2—1/4	12·0	—	—	—	—	90·0	3·9	1·3	27·7	8·7	74·1	100·0	10·3
	1/4—1/8	28·4	—	—	—	—								
	1/8—1/16	21·4	—	—	—	—	91·5	3·0	1·0		7·5	71·2	100·0	8·4
	1/16—1/32	16·7	—	—	—	—								
	1/32—1/64	10·1	—	—	—	—	—	—	—	—	—	—	100·0	11·3
	<1/64	11·4	—	—	—	—								
	1/2—0	100·0	—	—	—	—	90·7	3·4	1·2	27·7	8·1	72·9	100·0	9·5
Busty (2)	1 3/8—1/2	7·3	—	—	—	—	80·4	5·8	2·9	33·7	16·7	79·8	100·0	19·0
	1/2—1/4	21·1	—	—	—	—	91·9	4·0	1·8	32·4	6·3	80·4	100·0	9·3
	1/4—1/8	22·5	—	—	—	—	94·3	2·8	1·4	32·2	4·3	78·7	100·0	6·5
	1/8—1/16	17·1	—	—	—	—	94·5	2·5	1·2	33·6	4·3	77·8	100·0	6·1
	1/16—1/64	21·9	—	—	—	—								
	<1/64	10·1	—	—	—	—	—	—	—	—	—	—	100·0	8·8
	1 3/8—0	100·0	—	—	—	—	90·3	3·7	1·8	33·1	7·9	79·6	100·0	8·1
Busty (3)	1 3/8—1/2	11·1	—	—	—	—	88·7	3·7	3·0	40·1	8·3*	78·9	100·0	11·0
	1/2—1/4	21·7	—	—	—	—	91·9	3·6	1·7		6·4*		100·0	9·1
	1/4—1/8	23·0	—	—	—	—	94·4	3·0	1·2		4·4*		100·0	6·9
	1/8—1/16	12·2	—	—	—	—	92·0	2·5	1·9	40·0	6·1*	78·2	100·0	7·7
	1/16—1/64	15·2	—	—	—	—								
	<1/64	6·8	—	—	—	—	—	—	—	—	—	—	100·0	12·8
	1 3/8—0	100·0	—	—	—	—	91·7	3·2	2·0	40·1	6·3*	78·4	100·0	7·7

* >1·75 S.G.

Coal.	Size (in.).	Per cent. of size.	Per cent. by weight and per cent. ash in fractions at different specific gravities.											
			< 1·35 S.G.		1·35—1·40 S.G.		1·40—1·50 S.G.		1·50—1·60 S.G.		> 1·60 S.G.		Total.	
			Weight.	Ash.	Weight.	Ash.	Weight.	Ash.	Weight.	Ash.	Weight.	Ash.	Weight.	Ash.
Townley	$1\frac{1}{2}$—$\frac{1}{2}$	100·0	—	—	—	—	76·3	6·0	1·3	30·2	22·4	74·4	100·0	21·7
	$\frac{1}{2}$—$\frac{1}{4}$	6·7	—	—	—	—								
	$\frac{1}{4}$—$\frac{1}{8}$	22·3	—	—	—	—	94·3	4·0	1·1		4·6	65·4	100·0	7·2
	$\frac{1}{8}$—$\frac{1}{16}$	22·7	—	—	—	—				30·2				
	$\frac{1}{16}$—$\frac{1}{32}$	21·3	—	—	—	—	94·6	2·7	0·8		4·6	65·4	100·0	5·8
	$\frac{1}{32}$—$\frac{1}{64}$	14·2	—	—	—	—								
	$<\frac{1}{64}$	12·8	—	—	—	—	—	—	—	—	—	—	100·0	7·8
	$\frac{1}{2}$—0	100·0	—	—	—	—	94·4	3·3	1·0	30·2	4·6	65·4	100·0	6·5
"L"	$>\frac{1}{4}$	20·7	75·6	2·3	3·4	12·3	4·1	18·9	1·8	28·2	15·1	63·3	100·0	13·0
	$\frac{1}{4}$—$\frac{1}{8}$	26·0	76·7	2·2	3·0		3·7		1·9		14·7	64·5	100·0	12·8
	$\frac{1}{8}$—$\frac{1}{16}$	16·7	76·3	2·0	2·6	11·2	3·9	17·1	1·8	27·9	15·4	63·5	100·0	12·8
	$\frac{1}{16}$—$\frac{1}{64}$	21·9	73·2	1·7	3·1		3·6		1·8		18·3	64·7	100·0	14·6
	$<\frac{1}{64}$	14·7	—	—	—	—	—	—	—	—	—	—	100·0	15·9
	$>\frac{1}{4}$—0	100·0	75·4	2·1	3·0	11·8	3·8	18·0	1·8	28·1	15·9	64·0	100·0	13·7
"S"	$>\frac{1}{8}$	48·4	79·6	1·6	2·7	10·3	3·3	16·5	1·5	26·5	12·9	67·3	100·0	11·2
	$\frac{1}{8}$—$\frac{1}{64}$	31·0	76·8	1·3	2·7	10·6	3·1	15·9	1·4	25·4	16·0	67·3	100·0	12·9
	$<\frac{1}{64}$	20·6	—	—	—	—	—	—	—	—	—	—	100·0	13·3
	$>\frac{1}{8}$—0	100·0	78·1	1·5	2·7	10·4	3·2	16·2	1·5	25·9	14·5	67·0	100·0	12·2

NORTHUMBERLAND. D.C.B.	$2\frac{1}{2}$—2	27·7	85·0	2·2	1·5	17·0	3·3	17·5	1·0	34·0	9·2	66·7	100·0	12·6
	2—$1\frac{1}{2}$	11·4	85·5	2·0	2·2	10·3	1·2	17·9	1·4	29·7	9·7	70·3	100·0	14·0
	$1\frac{1}{2}$—1	12·3	86·0	1·8	2·0	13·7	1·3	19·9	1·7	22·8	9·0	60·6	100·0	14·2
	1—$\frac{3}{4}$	11·8	76·0	2·4	2·1	11·9	4·2	19·8	1·7	30·5	16·0	56·6	100·0	18·5
	$\frac{3}{4}$—$\frac{1}{2}$	6·1	79·4	2·8	1·5	11·4	3·5	18·8	1·7	29·1	13·9	56·7	100·0	20·4
	$\frac{1}{2}$—$\frac{1}{4}$	6·8	72·9	3·1	2·3	11·1	2·9	16·7	1·9	26·2	20·0	62·4	100·0	24·3
	$\frac{1}{4}$—$\frac{1}{8}$	10·3	51·7	2·4	5·8	11·4	5·5	26·5	2·0	26·0	35·0	61·9	100·0	25·5
	$\frac{1}{8}$—$\frac{1}{16}$	6·3	57·6	4·1	3·8	9·8	4·0	16·1	2·0	26·0	32·6	62·9	100·0	24·5
	$<\frac{1}{16}$	7·3	42·4	3·8	10·6	8·0	8·6	18·9	4·7	42·8	33·7	60·5	100·0	31·7
	$2\frac{1}{2}$—0	100·0	70·7	2·6	3·5	10·4	3·8	19·6	2·1	31·8	19·9	61·6	100·0	17·6
"C"	$2\frac{1}{2}$—2	29·1	74·5	3·1	1·8	11·8	2·2	18·4	1·2	30·0	20·3	50·1	100·0	17·5
	2—$1\frac{1}{2}$	13·4	75·0	3·5	2·2	11·8	0·5	23·7	2·2	32·1	20·1	57·0	100·0	17·4
	$1\frac{1}{2}$—1	10·5	73·4	2·8	1·1	10·1	0·9	22·2	2·5	23·8	22·1	65·6	100·0	17·1
	1—$\frac{3}{4}$	8·9	72·0	2·7	1·7	12·5	2·1	20·1	2·9	28·7	21·3	62·7	100·0	19·9
	$\frac{3}{4}$—$\frac{1}{2}$	7·4	67·7	2·5	2·8	13·8	2·1	19·6	2·4	29·6	25·0	62·4	100·0	22·3
	$\frac{1}{2}$—$\frac{1}{4}$	5·4	56·4	2·7	5·6	15·0	2·9	20·6	1·9	19·4	33·6	68·0	100·0	20·8
	$\frac{1}{4}$—$\frac{1}{8}$	12·1	51·5	3·0	3·0	17·0	4·8	19·7	2·0	16·0	38·7	68·7	100·0	26·7
	$\frac{1}{8}$—$\frac{1}{16}$	4·6	54·6	2·7	5·0	7·4	1·9	37·9	1·3	46·9	37·2	70·4	100·0	27·6
	$<\frac{1}{16}$	8·6	40·9	3·1	5·6	8·9	9·5	15·0	2·6	47·9	41·4	68·8	100·0	32·2
	$2\frac{1}{2}$—0	100·0	62·9	2·9	3·2	11·4	3·0	19·5	2·1	30·3	28·8	65·2	100·0	21·0
"A"	1—$\frac{1}{2}$	12·9	69·8	3·0	·4	13·5	3·6	22·1	3·2	31·9	21·0	64·8	100·0	17·8
	$\frac{1}{2}$—$\frac{1}{4}$	35·7	67·4	3·1	3·5		3·9		2·6		22·6	67·9	100·0	19·5
	$\frac{1}{4}$—$\frac{1}{8}$	17·5	59·2	2·7	3·1	11·1	4·3	20·2	3·4	31·0	30·0	66·0	100·0	23·7
	$\frac{1}{8}$—$\frac{1}{16}$	12·3	51·8	2·3	5·2		4·7		2·9		35·4	67·3	100·0	27·5
	$\frac{1}{16}$—$\frac{1}{64}$	13·9	49·2	2·4	3·4	10·6	4·8	17·6	4·0	30·4	38·6	66·8	100·0	29·3
	$<\frac{1}{64}$	7·7	—	—	—	—	—	—	—	—	—	—	100·0	31·2
	1—0	100·0	61·4	2·7	3·4	11·9	4·2	20·8	3·1	31·2	27·9	66·7	100·0	23·3

Coal.	Size (in.).	Per cent. of size.	< 1·35 S.G. Weight.	< 1·35 S.G. Ash.	1·30—1·40 S.G. Weight.	1·30—1·40 S.G. Ash.	1·40—1·50 S.G. Weight.	1·40—1·50 S.G. Ash.	1·50—1·60 S.G. Weight.	1·50—1·60 S.G. Ash.	> 1·60 S.G. Weight.	> 1·60 S.G. Ash.	Total. Weight.	Total. Ash.
			Per cent. by weight and per cent. ash in fractions at different specific gravities.											
"BC"	> 1/2	11·0	79·7	2·3	2·3	12·7	2·6	19·0	1·3	27·8	14·1	61·8	100·0	11·8
	1/2—1/4	37·0	77·7	2·2	2·5		2·3		1·9		15·6	63·2	100·0	12·9
	1/4—1/8	19·4	73·1	1·9	2·1	11·9	2·9	18·6	1·6	28·4	20·3	62·8	100·0	15·4
	1/8—1/16	11·4	67·6	1·9	2·0		3·0		1·4		26·0	63·6	100·0	19·2
	1/16—1/64	12·8	61·1	1·9	3·3	10·3	3·3	16·5	1·4	25·8	30·9	63·8	100·0	22·1
	< 1/64	8·4	—	—	—	—	—	—	—	—	—	—	100·0	24·2
	> 1/2—0	100·0	71·8	2·1	2·5	11·7	2·8	18·6	1·5	27·7	21·4	63·2	100·0	16·1
"B"	1—1/2	14·6	—	—	—	—	82·0	2·9	2·6	37·3	15·4*	70·8	100·0	14·2
	1/2—1/4	35·7	—	—	—	—	75·9	3·1	3·8	36·7	20·3*	70·5	100·0	18·0
	1/4—1/8	19·2	—	—	—	—	69·5	3·5	3·2		27·3*		100·0	22·9
	1/8—1/64	21·7	—	—	—	—	63·8	3·1	3·8	35·6	32·4*	66·2	100·0	24·8
	< 1/64	8·8	—	—	—	—	—	—	—	—	—	—	100·0	26·0
	1—0	100·0	—	—	—	—	72·8	3·1	3·4	36·7	23·8*	68·9	100·0	20·7
YORKSHIRE. Barnsley	> 3/8	56·0	81·1	2·5	4·6	11·3	3·5	16·0	1·5	20·3	9·3	70·0	100·0	9·9
	3/8—1/8	26·0	78·3	2·2	4·0	12·5	3·0	17·2	2·2	28·6	12·5	76·4	100·0	12·9
	1/8—1/25	12·4	73·3	1·6	3·3	11·0	3·0	15·2	1·7	25·0	18·7	74·3	100·0	16·3
	< 1/25	5·6	51·0	1·3	6·0	9·7	6·0	11·3	6·0	14·6	31·0	58·0	100·0	20·8
	> 3/8—0	100·0	70·9	2·0	4·5	11·0	3·9	14·3	2·9	19·6	17·8	67·0	100·0	12·1
Haigh Moor (1)	> 3/8	41·8	87·7	1·5	1·3	12·6	—	—	—	—	11·0	66·4	100·0	8·8
	3/8—1/8	27·4	78·8	1·8	1·0	10·0	1·0	16·4	—	—	19·2	82·0	100·0	17·4
	1/8—1/25	20·0	69·4	1·4	2·3	10·0	1·3	15·7	0·7	20·1	26·3	79·0	100·0	22·3
	< 1/25	10·8	53·0	1·7	8·0	9·6	6·0	11·7	5·0	17·7	28·0	66·5	100·0	21·8
	> 3/8—0	100·0	70·6	1·6	3·1	10·0	2·8	12·9	2·8	18·0	20·7	73·9	100·0	15·3

Shafton	$>\frac{3}{8}$	33·4	70·3	5·2	5·3	13·8	6·0	18·2	1·3	28·5	17·1	64·5	100·0	16·9
	$\frac{3}{8}$—$\frac{1}{8}$	29·6	74·0	4·8	3·0	12·7	5·6	16·5	1·8	24·2	15·6	62·0	100·0	15·0
	$\frac{1}{8}$—$\frac{1}{25}$	21·4	71·7	4·2	2·0	10·6	6·3	14·6	3·3	23·4	16·7	62·1	100·0	15·3
	$<\frac{1}{25}$	15·6	50·0	3·0	7·0	8·0	9·0	12·4	5·0	19·2	29·0	55·0	100·0	20·1
	$>\frac{3}{8}$—0	100·0	66·5	4·3	4·3	10·9	6·7	14·9	2·9	22·1	19·6	59·9	100·0	16·5
High Hazel	>1	10·2	100·0	1·8	—	—	—	—	—	—	—	—	100·0	1·8
	1—$\frac{3}{8}$	53·9	93·0	1·6	1·2	13·5	1·4	26·5	0·0	—	4·4	81·5	100·0	5·6
	$\frac{3}{8}$—$\frac{1}{8}$	26·8	89·3	1·9	0·6	11·5	1·0	17·2	0·3	27·2	8·8	78·8	100·0	9·0
	$\frac{1}{8}$—$\frac{1}{25}$	5·3	79·7	1·9	0·6	10·3	1·3	14·0	0·6	25·5	17·8	77·6	100·0	15·7
	$\frac{1}{25}$—$\frac{1}{50}$	1·6	67·0	1·5	4·4	8·9	3·3	13·7	1·1	21·8	24·2	73·5	100·0	19·9
	$<\frac{1}{50}$	2·2	34·6	1·7	9·9	7·8	9·5	8·4	11·8	14·8	34·6	51·1	100·0	21·6
	>1—0	100·0	73·8	1·7	3·3	8·6	3·2	11·9	2·7	16·1	17·0	66·7	100·0	7·2
Silkstone	>1	9·7	89·4	3·0	2·1	11·6	—	0·0	0·0	—	8·5	40·5	100·0	6·4
	1—$\frac{3}{8}$	28·8	87·8	2·2	2·2	10·3	1·3	18·0	1·2	22·7	7·5	72·5	100·0	8·1
	$\frac{3}{8}$—$\frac{1}{8}$	28·4	80·8	2·7	2·0	11·2	1·8	17·6	0·8	26·4	14·6	78·7	100·0	14·4
	$\frac{1}{8}$—$\frac{1}{25}$	21·3	67·7	2·1	2·3	10·0	2·2	15·2	1·3	25·0	26·5	72·0	100·0	21·5
	$<\frac{1}{25}$	11·8	54·0	2·3	10·0	9·6	5·0	15·7	5·0	24·0	26·0	69·6	100·0	22·3
	>1—0	100·0	75·9	2·5	3·7	10·1	2·1	16·2	1·7	24·3	16·6	69·3	100·0	14·3
"A"	>1	15·1	55·3	5·6	22·5	10·2	1·7	12·2	0·0	—	20·5	69·1	100·0	19·8
	1—$\frac{3}{8}$	22·4	45·8	5·2	11·4	8·8	9·3	14·1	2·9	19·7	27·6	70·6	100·0	25·0
	$\frac{3}{8}$—$\frac{1}{8}$	29·9	53·2	5·8	11·2	8·8	7·0	14·6	2·8	23·1	25·8	69·5	100·0	23·7
	$\frac{1}{8}$—$\frac{1}{25}$	21·9	49·3	5·0	7·3	9·1	6·0	15·1	3·7	22·0	33·7	71·6	100·0	29·0
	$<\frac{1}{25}$	10·7	33·0	5·6	7·0	8·4	3·0	10·5	4·0	22·1	53·0	67·3	100·0	39·3
	>1—0	100·0	47·3	5·4	12·5	9·3	5·4	13·9	2·7	21·8	32·1	69·3	100·0	26·3
					> 1·75 S.G.									

C.C.

Coal.	Size (in.).	Per cent. of size.	Per cent. by weight and per cent. ash in fractions at different specific gravities.											
			< 1·35 S.G.		1·35—1·40 S.G.		1·40—1·50 S.G.		1·50—1·60 S.G.		> 1·60 S.G.		Total.	
			Weight.	Ash.	Weight.	Ash.	Weight.	Ash.	Weight.	Ash.	Weight.	Ash.	Weight.	Ash.
Flockton	> 1	27·0	53·3	3·2	7·4	11·5	3·2	22·2	0·0	—	36·1	65·0	100·0	26·7
	1—$\frac{3}{8}$	23·4	70·0	4·0	4·0	13·1	4·0	20·7	5·0	24·3	17·0	72·4	100·0	17·7
	$\frac{3}{8}$—$\frac{1}{8}$	24·0	67·0	3·9	4·8	11·4	4·8	19·0	3·0	29·3	20·4	71·4	100·0	19·5
	$\frac{1}{8}$—$\frac{1}{25}$	16·3	52·7	3·1	5·3	10·2	5·7	16·8	2·3	30·3	34·0	74·0	100·0	28·9
	< $\frac{1}{25}$	9·3	33·7	3·3	5·4	8·0	3·0	13·1	2·5	20·3	57·0	63·0	100·0	38·3
	> 1—0	100·0	55·2	3·6	5·3	10·7	4·1	18·4	2·6	25·8	32·8	67·8	100·0	24·3
"B"	> $\frac{1}{25}$	90·6	79·8	1·4	—	—	2·1	14·7	1·4	33·7	16·7	86·8	100·0	14·7
	< $\frac{1}{25}$	9·4	—	—	—	—	—	—	—	—	—	—	100·0	20·3
Haigh Moor (2)	> $\frac{1}{4}$	30·0	87·7	0·9	0·7	11·1	0·6	19·2	0·5	27·3	10·5	77·3	100·0	9·3
	$\frac{1}{4}$—$\frac{1}{8}$	20·8	83·2	0·9	0·8	10·7	0·8	16·6	0·5	26·0	14·7	74·8	100·0	12·1
	$\frac{1}{8}$—$\frac{1}{16}$	15·9	74·2	0·9	0·9	9·2	1·3	13·4	0·6	25·2	23·0	72·8	100·0	17·8
	$\frac{1}{16}$—$\frac{1}{64}$	21·2	68·2	0·8	1·6	7·5	1·3	12·4	0·9	21·8	28·0	71·9	100·0	21·3
	< $\frac{1}{64}$	12·1	—	—	—	—	—	—	—	—	—	—	100·0	20·0
	> $\frac{1}{4}$—0	100·0	78·3	0·9	1·0	9·1	1·0	14·6	0·6	24·5	19·1	73·9	100·0	15·1
"K"	> 1	8·8	82·7	2·1	0·3	9·5	7·7	14·7	2·7	25·6	6·6	44·2	100·0	6·6
	1—$\frac{1}{2}$	19·8	85·1	2·5	2·1		4·1		2·2		6·5	50·9	100·0	6·9
	$\frac{1}{2}$—$\frac{1}{4}$	26·2	83·3	2·0	1·1	9·2	5·7	13·9	1·0	24·9	8·9	67·2	100·0	8·9
	$\frac{1}{4}$—$\frac{1}{8}$	14·8	76·8	1·9	2·0		5·6		1·4		14·2	67·3	100·0	12·3
	$\frac{1}{8}$—$\frac{1}{16}$	10·6	70·8	1·6	1·9	7·6	6·1	12·9	1·6	25·3	19·6	71·9	100·0	16·6
	$\frac{1}{16}$—$\frac{1}{64}$	12·0	61·2	1·6	2·6		7·8		2·2		26·8	72·9	100·0	20·4
	< $\frac{1}{64}$	7·8	—	—	—	—	—	—	—	—	—	—	100·0	20·3
	> 1—0	100·0	76·6	2·0	1·6	8·5	6·2	13·8	1·8	25·3	13·8	67·2	100·0	11·9

DERBYSHIRE. Top Hard .	$\frac{1}{2}$—$\frac{1}{4}$	32·8	90·0	2·2	0·6	13·4	1·7	18·0	0·6	32·1	7·1	67·8	100·0	7·8
	$\frac{1}{4}$—$\frac{1}{8}$	24·4	88·4	2·1	1·2	12·5	2·1	18·9	0·8	31·9	7·5	64·7	100·0	8·7
	$\frac{1}{8}$—$\frac{1}{64}$	34·1	77·8	1·9	1·8	10·0	3·0	18·0	1·2	28·2	16·2	72·4	100·0	14·3
	$<\frac{1}{64}$	8·7	—	—	—	—	—	—	—	—	—	—	100·0	15·2
	$\frac{1}{2}$—0	100·0	85·4	2·1	1·2	11·9	2·3	18·3	0·8	30·1	10·3	69·4	100·0	10·9
Deep Hard .	$\frac{1}{2}$—$\frac{1}{4}$	31·5	85·4	4·0	5·5	13·3	2·4	19·2	0·8	31·9	5·9	62·3	100·0	8·6
	$\frac{1}{4}$—$\frac{1}{8}$	25·8	82·9	3·7	4·9	12·4	3·6	17·8	1·9	29·6	6·7	65·2	100·0	9·7
	$\frac{1}{8}$—$\frac{1}{64}$	34·2	76·8	3·0	6·5	11·4	5·2	17·5	1·5	26·2	10·0	66·7	100·0	12·8
	$<\frac{1}{64}$	8·5	—	—	—	—	—	—	—	—	—	—	100·0	14·9
	$\frac{1}{2}$—0	100·0	81·7	3·6	5·7	12·4	3·7	18·1	1·4	29·2	7·5	65·2	100·0	10·8
Waterloo . .	$\frac{1}{2}$—$\frac{1}{4}$	37·6	79·8	1·6	2·1	11·0	4·5	16·0	0·8	21·1	12·8	73·4	100·0	12·1
	$\frac{1}{4}$—$\frac{1}{8}$	23·9	81·4	1·6	3·0	10·1	4·0	15·8	1·3	26·5	10·3	73·5	100·0	9·9
	$\frac{1}{8}$—$\frac{1}{64}$	30·7	78·8	1·6	3·0	10·2	3·8	15·9	1·8	24·4	12·6	72·3	100·0	12·2
	$<\frac{1}{64}$	7·8	—	—	—	—	—	—	—	—	—	—	100·0	15·2
	$\frac{1}{2}$—0	100·0	80·0	1·6	2·7	10·1	4·1	15·9	1·3	24·4	11·9	73·1	100·0	11·8
NOTTINGHAM-SHIRE. "O." . .	$>\frac{1}{2}$	35·2	84·8	3·6	·3	11·3	4·8	19·1	0·4	32·1	2·7	68·6	100·0	6·8
	$\frac{1}{2}$—$\frac{1}{4}$	22·7	84·6	3·5	4·4	12·3	4·6	18·2	0·8	30·1	5·8	57·5	100·0	8·7
	$\frac{1}{4}$—$\frac{1}{8}$	21·6	80·6	3·2	3·9		5·4		1·1		9·0	60·7	100·0	11·2
	$\frac{1}{8}$—$\frac{1}{16}$	10·5	75·0	3·1	4·0	11·1	6·6	17·1	1·5	29·9	12·9	78·1	100·0	14·4
	$\frac{1}{16}$—$\frac{1}{64}$	6·3	64·5	3·1	5·8		7·3		1·7		20·7	79·2	100·0	20·9
	$<\frac{1}{64}$	3·7	—	—	—	—	—	—	—	—	—	—	100·0	24·1
	$>\frac{1}{2}$—0	100·0	77·9	3·4	5·1	11·5	5·7	17·8	1·1	30·1	10·2	72·6	100·0	10·5

Coal.	Size (in.).	Per cent. of size.	Per cent. by weight and per cent. ash in fractions at different specific gravities.											
			> 1·35 S.G.		1·35—1·40 S.G.		1·40—1·50 S.G.		1·50—1·60 S.G.		> 1·60 S.G.		Total.	
			Weight.	Ash.	Weight.	Ash.	Weight.	Ash.	Weight.	Ash.	Weight.	Ash.	Weight.	Ash.
"D"	$\frac{3}{16}$—$\frac{1}{16}$	32·2	51·0	3·9	—	—	7·9	15·2	2·5	26·2	38·6	75·4	100·0	32·9
	$\frac{1}{16}$—$\frac{1}{64}$	41·8	38·6	3·7	—	—	10·5	12·5	2·5	25·1	48·4	75·9	100·0	40·2
	$<\frac{1}{64}$	26·0	—	—	—	—	—	—	—	—	—	—	100·0	34·6
	$\frac{1}{16}$—0	100·0	44·8	3·8	—	—	9·2	13·7	2·5	25·6	43·5	75·7	100·0	36·3
LANCASHIRE. Upper Mountain Mine.	$\frac{1}{2}$—$\frac{1}{4}$	11·4	66·6	3·7	9·1	14·4	12·8	20·7	4·8	32·1	6·7	52·7	100·0	11·5
	$\frac{1}{4}$—$\frac{1}{8}$	32·6	64·7	3·5	9·9	14·2	11·5	21·4	5·4		8·5	53·2	100·0	12·4
	$\frac{1}{8}$—$\frac{1}{16}$	19·7	69·2	3·1	7·5	14·0	5·3	21·1	5·3	31·6	12·7	57·8	100·0	13·4
	$\frac{1}{16}$—$\frac{1}{64}$	22·2	72·0	2·8	7·0	14·6	8·0	20·7	4·6		8·4	55·9	100·0	10·9
	$<\frac{1}{64}$	14·1	—	—	—	—	—	—	—	—	—	—	100·0	11·0
	$\frac{1}{2}$—0	100·0	68·1	3·3	8·4	14·4	9·4	21·0	5·0	31·9	9·1	55·5	100·0	12·0
Wigan 7 ft.	$>\frac{1}{2}$	36·3	—	—	—	—	83·3	2·3	0·5	36·0	16·2	67·7	100·0	13·0
	$\frac{1}{2}$—$\frac{1}{4}$	28·8	—	—	—	—	87·3	2·0	0·9	31·8	11·8	78·3	100·0	11·3
	$\frac{1}{4}$—$\frac{1}{8}$	15·2	—	—	—	—	85·0	2·1	1·0		14·0	76·7	100·0	12·9
	$\frac{1}{8}$—$\frac{1}{16}$	6·5	—	—	—	—	82·8	2·2	1·1	29·0	16·1	78·2	100·0	14·7
	$\frac{1}{16}$—$\frac{1}{64}$	8·4	—	—	—	—	77·8	2·3	1·2		21·0	78·2	100·0	18·6
	$<\frac{1}{64}$	4·8	—	—	—	—	—	—	—	—	—	—	100·0	20·8
	$>\frac{1}{2}$—0	100·0	—	—	—	—	83·2	2·2	1·0	30·9	15·8	75·6	100·0	13·4
Wigan 4 ft.	$>\frac{1}{2}$	32·8	—	—	—	—	80·2	5·5	2·5	27·0	17·3	69·6	100·0	17·1
	$\frac{1}{2}$—$\frac{1}{4}$	37·0	—	—	—	—	83·6	4·9	2·2	27·5	14·4	70·8	100·0	14·9
	$\frac{1}{4}$—$\frac{1}{8}$	15·8	—	—	—	—	79·7	4·6	1·9		18·4	72·7	100·0	17·5
	$\frac{1}{8}$—$\frac{1}{16}$	6·4	—	—	—	—	77·8	4·4	2·0	26·9	20·2	76·4	100·0	19·3
	$\frac{1}{16}$—$\frac{1}{64}$	5·1	—	—	—	—	74·6	4·3	2·0		23·4	76·6	100·0	21·5
	$<\frac{1}{64}$	2·9	—	—	—	—	—	—	—	—	—	—	100·0	17·1
	$>\frac{1}{2}$—0	100·0	—	—	—	—	79·2	4·6	2·1	27·2	18·7	73·6	100·0	16·7

Potato Delf	$\frac{1}{2}$—$\frac{1}{4}$	35·9	79·7	2·7	—	—	7·2	14·4	2·1	25·9	11·0	61·7	100·0	9·8
	$\frac{1}{4}$—$\frac{1}{8}$	19·6	73·2	2·5	—	—	7·3	14·0	2·0	26·0	17·5	66·9	100·0	15·2
	$\frac{1}{8}$—$\frac{1}{16}$	15·6	67·1	2·2	—	—	6·9	12·6	1·9	25·8	24·1	65·5	100·0	19·2
	$\frac{1}{16}$—$\frac{1}{32}$	10·7	60·8	2·1	—	—	7·0	11·5	2·6	24·9	29·6	67·3	100·0	22·0
	$<\frac{1}{32}$	18·2	—	—	—	—	—	—	—	—	—	—	100·0	22·6
	$\frac{1}{2}$—0	100·0	70·2	2·4	—	—	7·1	13·1	2·1	25·6	20·6	66·0	100·0	16·0
Higher and Lower Florida.	$\frac{1}{2}$—$\frac{1}{4}$	33·5	69·7	1·5	—	—	2·1	13·7	0·6	24·4	7·6	70·5	100·0	7·6
	$\frac{1}{4}$—$\frac{1}{8}$	21·4	84·2	1·4	—	—	2·4	12·3	0·7	24·4	12·7	71·8	100·0	10·5
	$\frac{1}{8}$—$\frac{1}{16}$	16·2	80·3	1·3	—	—	2·8	10·6	0·7	21·5	16·2	69·0	100·0	13·5
	$\frac{1}{16}$—$\frac{1}{32}$	10·7	76·9	1·3	—	—	2·6		0·9		19·6	66·5	100·0	14·3
	$<\frac{1}{32}$	18·2	—	—	—	—	—	—	—	—	—	—	100·0	15·4
	$\frac{1}{2}$—0	100·0	82·8	1·4	—	—	2·5	11·8	0·7	22·9	14·0	68·9	100·0	11·3
CUMBERLAND. "R"	$>\frac{1}{8}$	15·8	46·8	3·7	2·5		1·2		1·0		48·5	85·3	100·0	43·8
	$\frac{1}{8}$—$\frac{1}{16}$	24·7				9·5		14·4		23·2				
	$\frac{1}{16}$—$\frac{1}{64}$	32·8	45·7	3·3	2·7		2·7		0·9		48·0	85·2	100·0	43·3
	$<\frac{1}{64}$	26·7	—	—	—	—	—	—	—	—	—	—	100·0	33·5
	$>\frac{1}{8}$—0	100·0	46·2	3·5	2·6	9·5	2·0	14·4	1·0	23·2	48·2	85·3	100·0	40·8
"O"	$\frac{1}{10}$—$\frac{1}{20}$	100·0	71·4	3·3	··1	9 8	4·0	15·9	1·5	26·4	18·0	71·7	100·0	16·8
"H"	$>\frac{3}{8}$	42·9	48·0	4·9	—	—	—	—	30·0	15·6	22·0	61·2	100·0	20·4
	$\frac{3}{8}$—$\frac{1}{8}$	28·6	51·2	3·4	—	—	—	—	17·8	16·5	31·0	—	100·0	23·7
	$<\frac{1}{8}$	28·5	31·0	3·9	—	—	—	—	18·0	17·3	51·0	—	100·0	38·4
	$>\frac{3}{8}$—0	100·0	43·4	4·1	—	—	—	—	21·9	16·3	34·7	61·2	100·0	25·6

Coal.	Size (in.).	Per cent. of size.	< 1·35 S.G. Weight.	< 1·35 S.G. Ash.	1·35—1·40 S.G. Weight.	1·35—1·40 S.G. Ash.	1·40—1·50 S.G. Weight.	1·40—1·50 S.G. Ash.	1·50—1·60 S.G. Weight.	1·50—1·60 S.G. Ash.	> 1·60 S.G. Weight.	> 1·60 S.G. Ash.	Total. Weight.	Total. Ash.
			Per cent. by weight and per cent. ash in fractions at different specific gravities.											
NORTH STAFFORDSHIRE. "S"	$\frac{1}{2}$—$\frac{1}{4}$	20·4	63·1	2·4	4·3	9·5	3·7	16·8	1·0	26·2	27·9	78·9	100·0	24·8
	$\frac{1}{4}$—$\frac{1}{8}$	20·5	61·6	1·9	3·1		4·0		1·2		30·1	77·9	100·0	25·8
	$\frac{1}{8}$—$\frac{1}{16}$	17·9	60·2	1·8	3·4	9·1	4·2	17·5	1·6	28·0	30·6	78·5	100·0	26·7
	$\frac{1}{16}$—$\frac{1}{32}$	14·2	58·3	1·7	3·2		4·7		2·5		31·3	77·6	100·0	27·1
	$\frac{1}{32}$—$\frac{1}{64}$	10·4	56·6	1·6	—	—	—	—	11·1	17·9	32·3	78·5	100·0	28·3
	< $\frac{1}{64}$	16·4	—	—	—	—	—	—	—	—	—	—	100·0	27·4
	$\frac{1}{2}$—0	100·0	59·9	1·9	3·5	9·3	4·3	17·2	1·8	27·4	30·5	78·3	100·0	26·4
SOUTH STAFFORDSHIRE. "B"	> $\frac{3}{4}$	100·0	66·9	4·1	—	—	10·2	19·9	1·5	30·9	21·4	71·4	100·0	20·5
	$\frac{3}{4}$—$\frac{1}{8}$	57·7	61·4	3·9	—	—	9·6	17·7	2·5	28·5	26·5	68·8	100·0	23·1
	$\frac{1}{8}$—$\frac{1}{64}$	30·6	44·3	3·6	—	—	10·0	16·0	3·1	29·7	42·6	70·2	100·0	34·1
	< $\frac{1}{64}$	11·7	—	—	—	—	—	—	—	—	—	—	100·0	35·8
	$\frac{3}{4}$—0	100·0	52·9	3·8	—	—	9·8	16·8	2·8	29·1	34·5	69·5	100·0	27·9
"R"	1—$\frac{1}{2}$	22·7	80·0	2·6	2·8	12·0	5·2	20·1	1·9	30·4	10·2	56·7	100·0	10·0
	$\frac{1}{2}$—$\frac{1}{4}$	27·0	79·1	2·5	3·1		4·3		1·5		12·0	65·2	100·0	11·5
	$\frac{1}{4}$—$\frac{1}{8}$	15·1	76·4	2·7	4·7	9·7	5·0	17·6	1·5	29·1	12·4	66·6	100·0	12·1
	$\frac{1}{8}$—$\frac{1}{16}$	12·4	73·9	2·5	5·0		6·8		1·3		13·0	68·6	100·0	12·8
	$\frac{1}{16}$—$\frac{1}{64}$	14·6	68·4	2·7	4·7	9·0	8·5	14·5	1·7	26·4	16·7	68·9	100·0	15·5
	< $\frac{1}{64}$	8·2	—	—	—	—	—	—	—	—	—	—	100·0	19·4
	1—0	100·0	75·5	2·6	4·1	10·2	6·0	17·5	1·5	29·1	12·9	65·9	100·0	12·6.

WARWICK-SHIRE. "B"	>½	27·5	91·5	1·3	2·1	5·4	1·2	8·7	0·8	22·1	4·4	71·9	100·0	4·7
	½—¼	27·6	88·0	1·2	1·9		2·9		0·7		6·5	64·8	100·0	5·8
	¼—⅛	14·5	85·6	1·3	0·4	5·1	4·1	8·5	1·0	21·1	8·9	66·0	100·0	7·5
	⅛—$\frac{1}{16}$	11·4	82·2	1·2	1·0		5·2		1·0		10·6	66·8	100·0	8·8
	$\frac{1}{16}$—$\frac{1}{64}$	12·3	75·0	0·9	2·4	2·5	7·9	5·8	1·1	21·6	13·6	69·1	100·0	10·8
	<$\frac{1}{64}$	6·7	—	—	—	—	—	—	—	—	—	—	100·0	10·3
	>½—0	100·0	84·5	1·2	1·5	4·4	4·3	7·5	0·9	21·6	8·8	67·6	100·0	7·0
LEICESTER-SHIRE. "R"	1¼—½	35·0	—	—	—	—	80·5	3·0	1·6	29·1	17·9	73·7	100·0	15·9
	½—$\frac{1}{16}$	55·2	—	—	—	—	79·6	2·9	1·6	28·7	18·8	72·4	100·0	16·4
	<$\frac{1}{16}$	9·8	—	—	—	—	—	—	—	—	—	—	100·0	20·2
	1¼—0	100·0	—	—	—	—	80·0	2·9	1·6	28·9	18·4	73·0	100·0	16·6
GLOUCESTER-SHIRE. "B"	½—¼	23·0	49·8	4·1	9·2	8·4	9·0	14·7	3·2	23·2	28·7	70·3	100·0	25·2
	¼—⅛	23·4	56·2	3·9	7·5		8·3		3·5		24·5	71·5	100·0	22·4
	⅛—$\frac{1}{16}$	16·7	56·8	3·5	8·8	8·5	7·8	14·8	3·5	24·2	23·1	69·5	100·0	20·9
	$\frac{1}{16}$—$\frac{1}{64}$	22·5	51·5	3·4	10·9		6·0		3·9		27·7	70·9	100·0	24·2
	<$\frac{1}{64}$	14·4	—	—	—	—	—	—	—	—	—	—	100·0	29·0
	½—0	100·0	53·6	3·7	9·1	8·5	7·8	14·7	3·5	23·7	26·0	70·3	100·0	24·2
KENT. "A"	>¼	28·6	24·2	4·8	16·3	13·0	16·6	20·3	9·1	30·0	33·8	61·9	100·0	30·4
	¼—⅛	23·1	70·1	4·0	5·3		7·7		2·5		14·4	62·3	100·0	14·3
	⅛—$\frac{1}{16}$	15·5	80·4	3·4	3·2	13·0	4·6	19·3	1·8	29·6	10·0	61·3	100·0	10·7
	$\frac{1}{16}$—$\frac{1}{64}$	22·2	84·3	2·8	1·7		3·9		1·3		8·8	60·1	100·0	9·0
	<$\frac{1}{64}$	10·6	—	—	—	—	—	—	—	—	—	—	100·0	12·3
	>¼—0	100·0	64·7	3·5	6·6	13·0	8·2	20·0	3·7	29·9	16·8	61·8	100·0	17·0

Coal.	Size (in.).	Per cent. of size.	Per cent. by weight and per cent. ash in fractions at different specific gravities.											
			< 1·35 S.G. Weight.	< 1·35 S.G. Ash.	1·35—1·40 S.G. Weight.	1·35—1·40 S.G. Ash.	1·40—1·50 S.G. Weight.	1·40—1·50 S.G. Ash.	1·50—1·60 S.G. Weight.	1·50—1·60 S.G. Ash.	> 1·60 S.G. Weight.	> 1·60 S.G. Ash.	Total. Weight.	Total. Ash.
"B"	1—½	10·0	59·2	3·3	7·3	8·0	10·0	12·9	2·0	20·5	21·5	75·7	100·0	20·6
	½—¼	15·6	64·8	2·7	6·9		7·6		1·9		18·8	74·2	100·0	17·7
	¼—⅛	17·0	74·7	2·3	5·5	8·7	5·3	14·6	1·5	22·5	13·0	72·6	100·0	12·7
	⅛—1/16	18·2	80·3	2·1	4·8		4·3		1·1		9·7	73·1	100·0	10·0
	1/16—1/64	25·6	82·3	1·9	4·1	9·2	3·7	15·4	1·2	24·6	8·7	70·8	100·0	9·0
	< 1/64	13·6	—	—	—	—	—	—	—	—	—	—	100·0	12·6
	1—0	100·0	72·3	2·4	5·7	8·4	6·2	13·7	1·5	21·8	14·3	73·8	100·0	12·9
SCOTLAND. "P"	½—¼	16·4	46·3	4·0	0·5	10·5	12·0	17·9	5·2	31·4	36·0	67·6	100·0	30·0
	¼—⅛	23·6	57·2	3·8	0·5		13·9		4·4		24·0	68·2	100·0	22·4
	⅛—1/16	20·4	64·2	3·4	2·9	9·9	10·8	16·6	3·5	30·8	18·6	70·7	100·0	18·5
	1/16—1/64	25·8	71·0	2·9	1·6		10·6		2·1		14·7	70·0	100·0	14·9
	< 1/64	13·8	—	—	—	—	—	—	—	—	—	—	100·0	15·1
	½—0	100·0	59·7	3·5	1·4	10·1	11·8	17·3	3·8	31·2	23·3	68·8	100·0	19·9
"K"	1—½	35·1	93·9	3·2	0·2	10·8	4·3	15·5	0·7	29·6	0·9	62·8	100·0	4·5
	½—¼	31·5	86·6	3·1	0·4		6·1		1·2		5·7	65·8	100·0	7·7
	¼—⅛	13·5	70·7	3·2	0·3	9·5	5·9	14·2	1·2	27·0	21·9	75·8	100·0	20·1
	⅛—1/16	8·3	65·7	3·0	1·3		5·8		1·5		25·7	76·4	100·0	22·9
	1/16—1/64	7·6	61·6	2·7	1·1		8·0		2·0		27·3	76·8	100·0	23·5
	< 1/64	4·0	—	—	—	—	—	—	—	—	—	—	100·0	22·7
	1—0	100·0	75·7	3·0	0·7	9·7	6·0	14·6	1·3	27·6	16·3	75·4	100·0	11·3

"R"	$>\frac{1}{2}$	22·9	28·4	4·3	33·6	7·2	10·7	14·8	4·0	27·6	23·3	67·7	100·0	22·1
	$\frac{1}{2}-\frac{1}{8}$	44·0	24·5	3·5	54·2	7·0	8·0	17·6	1·6	27·5	11·7	65·0	100·0	14·1
	$\frac{1}{8}-\frac{1}{64}$	25·6	32·9	2·9	35·7	6·0	17·0	12·9	2·0	25·4	12·4	64·6	100·0	13·8
	$<\frac{1}{64}$	7·5	—	—	—	—	—	—	—	—	—	—	100·0	17·1
	$>\frac{1}{2}-0$	100·0	28·6	3·5	41·2	6·8	11·9	14·6	2·5	26·1	15·8	66·2	100·0	16·1
"W"	$>\frac{1}{2}$	25·2	76·7	4·3	3·3	14·1	2·1	23·7	1·4	32·8	16·5	75·2	100·0	17·3
	$\frac{1}{2}-\frac{1}{8}$	43·4	71·6	4·3	2·8	14·0	3·7	20·5	1·6	29·5	20·3	73·7	100·0	19·6
	$\frac{1}{8}-\frac{1}{64}$	23·2	59·9	3·7	3·5	10·8	4·5	16·4	3·1	27·0	29·0	74·7	100·0	25·8
	$<\frac{1}{64}$	8·2	—	—	—	—	—	—	—	—	—	—	100·0	29·1
	$>\frac{1}{2}-0$	100·0	69·4	4·1	3·2	12·9	3·4	19·3	2·1	29·0	21·9	74·4	100·0	21·3
"L"	$>\frac{1}{16}$	44·3	59·6	3·3	5·0	10·7	5·4	17·3	3·6	28·3	26·4	69·3	100·0	22·9
	$\frac{1}{16}-\frac{1}{64}$	34·0	45·6	3·4	10·8	8·3	8·0	14·7	4·2	25·8	31·4	68·1	100·0	26·0
	$<\frac{1}{64}$	21·7	—	—	—	—	—	—	—	—	—	—	100·0	28·9
	$>\frac{1}{16}-0$	100·0	52·6	3·3	7·9	9·1	6·7	15·8	3·9	26·9	28·9	68·6	100·0	25·3
SOUTH WALES. "W"	$\frac{1}{2}-\frac{1}{4}$	15·3	63·0	3·3	13·1	9·3	16·9	13·8	1·0	24·8	6·0	76·6	100·0	10·5
	$\frac{1}{4}-\frac{1}{8}$	17·8	66·9	2·8	12·7		13·8		1·3		5·3	65·2	100·0	9·2
	$\frac{1}{8}-\frac{1}{16}$	19·5	71·8	2·5	9·1	9·[illegible]	13·0	14·1	1·1	26·9	5·0	72·3	100·0	8·3
	$\frac{1}{16}-\frac{1}{64}$	28·8	76·4	2·2	3·6		9·0		1·0		5·0	71·8	100·0	7·6
	$<\frac{1}{64}$	18·6	—	—	—	—	—	—	—	—	—	—	100·0	8·7
	$\frac{1}{2}-0$	100·0	69·5	2·7	[illegible]·9	9·2	13·2	13·9	1·1	25·8	5·3	71·2	100·0	8·7
"T"	$1\frac{1}{8}-\frac{1}{2}$	13·3	57·7	2·4	10·1	7·3	4·6	16·9	2·1	27·2	25·5	73·2	100·0	22·1
	$\frac{1}{2}-\frac{1}{4}$	17·9	59·6	2·5	11·2		6·8		2·3		20·1	71·9	100·0	18·5
	$\frac{1}{4}-\frac{1}{8}$	16·6	64·6	2·3	10·4	6·5	5·4	15·4	2·2	26·4	17·4	71·8	100·0	16·0
	$\frac{1}{8}-\frac{1}{16}$	16·0	70·8	1·8	9·5		4·1		1·6		14·0	72·6	100·0	13·2
	$\frac{1}{16}-\frac{1}{64}$	22·6	73·4	1·5	9·1	6·1	3·5	13·9	1·5	25·3	12·5	72·7	100·0	11·6
	$<\frac{1}{64}$	13·6	—	—	—	—	—	—	—	—	—	—	100·0	9·5
	$1\frac{1}{8}-0$	100·0	65·2	2·1	10·0	6·8	4·9	15·8	2·0	26·6	17·9	72·3	100·0	14·9

Coal.	Size (in.).	Per cent. of size.	Per cent. by weight and per cent. ash in fractions at different specific gravities.											
			< 1·35 S.G.		1·35—1·40 S.G.		1·40—1·50 S.G.		1·50—1·60 S.G.		> 1·60 S.G.		Total.	
			Weight.	Ash.	Weight.	Ash.	Weight.	Ash.	Weight.	Ash.	Weight.	Ash.	Weight.	Ash.
"B"	$>\frac{1}{2}$	13·0	67·5	2·5	3·9	9·7	2·2	15·4	2·4	27·6	24·0	67·3	100·0	19·3
	$\frac{1}{2}$—$\frac{1}{8}$	25·7	76·8	1·9	3·5	8·9	3·2	15·3	1·3	22·3	15·2	71·6	100·0	13·4
	$\frac{1}{8}$—$\frac{1}{64}$	36·2	77·9	1·6	4·0	8·7	2·0	15·7	1·2	24·8	14·9	68·5	100·0	12·4
	$<\frac{1}{64}$	15·1	—	—	—	—	—	—	—	—	—	—	100·0	10·7
	$>\frac{1}{2}$—0	100·0	74·1	2·0	3·8	9·1	2·5	15·5	1·6	25·5	18·0	68·8	100·0	12·0
"M"	3—$1\frac{1}{2}$	100·0	31·4	3·2	41·6	6·8	13·0	15·9	3·6	27·6	10·0	58·2	100·0	12·8
	$1\frac{1}{2}$—$\frac{1}{2}$	56·8	37·1	3·2	24·6	6·9	12·0	13·9	4·5	25·8	21·8	59·8	100·0	18·7
	$\frac{1}{2}$—$\frac{1}{4}$	37·2	42·9	3·3	23·9	6·7	13·1	14·7	3·9	26·7	16·2	58·6	100·0	15·4
	$<\frac{1}{4}$	6·0	—	—	—	—	—	—	—	—	—	—	100·0	—
	$1\frac{1}{2}$—0	100·0	40·0	3·3	24·2	6·8	12·6	14·3	4·2	26·2	19·0	59·2	100·0	—
Anthracite (1)	$\frac{1}{4}$—$\frac{1}{8}$	19·0	—	—	1·5	0·5	76·0	1·2	3·7	12·9	18·8	70·1	100·0	14·6
	$\frac{1}{8}$—$\frac{1}{16}$	24·8	—	—	3·0	0·6	72·5	1·2	4·4	11·8	20·1	67·2	100·0	15·1
	$\frac{1}{16}$—$\frac{1}{64}$	33·4	—	—	6·5	0·7	69·1	1·5	5·3	12·9	19·1	63·6	100·0	13·8
	$<\frac{1}{64}$	22·8	—	—	—	—	—	—	—	—	—	—	100·0	12·7
	$\frac{1}{4}$—0	100·0	—	—	3·7	0·6	72·5	1·3	4·5	12·6	19·3	66·9	100·0	13·6
Anthracite (2)	$\frac{1}{8}$—$\frac{1}{16}$	10·0	0·0	—	52·0	1·2	32·0	4·0	2·0	17·6	14·0	83·6	100·0	13·8
	$\frac{1}{16}$—$\frac{1}{64}$	49·3	0·0	—	58·7	1·3	28·0	4·3	1·9	17·2	11·4	76·8	100·0	11·1
	$<\frac{1}{64}$	40·7	—	—	—	—	—	—	—	—	—	—	100·0	11·9
	$\frac{1}{8}$—0	100·0	0·0	—	55·4	1·3	30·0	4·1	1·9	17·4	12·7	80·5	100·0	11·7

Table II.—Summary of Float and Sink Results. British Coals

Coalfield.	Coal.	Size (in.).	"Pure Coal." $< 1\cdot35$ S.G. Weight.	Ash.	Middlings. $1\cdot35$—$1\cdot60$ S.G. Weight.	Ash.	Dirt. $> 1\cdot60$ S.G. Weight.	Ash.	Total. Weight.	Ash.
GREAT BRITAIN. DURHAM	Five Quarter	$>\frac{1}{4}$—$\frac{1}{64}$	70·9	2·4	14·5	14·7	14·6	68·8	100·0	13·9
	"T"	$>\frac{1}{2}$—$\frac{1}{64}$	74·4	2·4	7·6	14·8	18·0	68·2	100·0	15·2
	"W"	$>\frac{1}{2}$—$\frac{1}{64}$	74·5	2·6	13·3	17·1	12·2	62·7	100·0	11·9
	Brockwell	$\frac{1}{2}$—$\frac{1}{64}$	92·2*	4·5	1·5	28·2	6·3	67·8	100·0	8·8
	Busty (1)	$\frac{1}{2}$—$\frac{1}{64}$	90·7*	3·4	1·2	27·7	8·1	72·9	100·0	9·3
	Busty (2)	$1\frac{3}{8}$—$\frac{1}{64}$	90·3*	3·7	1·8	33·1	7·9	79·6	100·0	10·2
	Busty (3)	$1\frac{3}{8}$—$\frac{1}{64}$	91·7*	3·2	2·0	40·1	6·3†	78·4	100·0	8·7
	Townley	$\frac{1}{2}$—$\frac{1}{64}$	94·4*	3·3	1·0	30·2	4·6	65·4	100·0	6·4
	"L"	$>\frac{1}{4}$—$\frac{1}{64}$	75·4	2·1	8·7	18·0	15·9	64·0	100·0	13·3
	"S"	$>\frac{1}{8}$—$\frac{1}{64}$	78·1	1·5	7·4	16·0	14·5	67·0	100·0	12·0
	Average	—	83·3	2·9	5·9	24·0	10·8	69·5	100·0	11·0
NORTHUMBERLAND	D.C.B.	$2\frac{1}{2}$—0	70·7	2·6	9·4	18·9	19·9	61·6	100·0	15·9
	"C"	$2\frac{1}{2}$—0	62·9	2·9	8·3	19·2	28·8	65·2	100·0	22·2
	"A"	1—$\frac{1}{64}$	61·4	2·7	10·7	21·0	27·9	66·7	100·0	22·5
	"BC"	$\frac{1}{2}$—$\frac{1}{64}$	71·8	2·1	6·8	18·1	21·4	63·4	100·0	16·3
	"B"	1—$\frac{1}{64}$	72·8*	3·1	3·4	36·7	23·8†	68·9	100·0	20·0
	Average	—	67·9	2·7	7·7	22·8	24·4	65·2	100·0	19·4

* $< 1\cdot50$ S.G. † $> 1\cdot75$ S.G.

Coalfield.	Coal.	Size (in.).	"Pure Coal." <1·35 S.G. Weight.	Ash.	Middlings. 1·35—1·60 S.G. Weight.	Ash.	Dirt. > 1·60 S.G. Weight.	Ash.	Total. Weight.	Ash.
YORKSHIRE	Barnsley	$>\frac{3}{8}$—0	70·9	2·0	11·3	14·3	17·8	67·0	100·0	12·1
	Haigh Moor (1)	$>\frac{3}{8}$—0	70·6	1·6	8·7	9·7	20·7	78·9	100·0	15·3
	Shafton	$>\frac{3}{8}$—0	66·5	4·2	13·9	15·2	19·6	59·9	100·0	16·5
	High Hazel	$>\frac{3}{8}$—0	73·8	1·7	9·2	9·9	17·0	66·7	100·0	7·2
	Silkstone	$>\frac{3}{8}$—0	75·9	2·5	7·5	14·8	16·6	69·3	100·0	14·3
	Flockton	$>\frac{3}{8}$—0	55·2	3·6	12·0	16·5	32·8	67·8	100·0	24·3
	"A"	$>\frac{3}{8}$—0	47·3	5·4	20·6	12·1	32·1	69·3	100·0	26·3
	"B"	$>\frac{1}{25}$	79·8	1·4	3·5	22·3	16·7	86·8	100·0	16·5
	Haigh Moor (2)	$>\frac{1}{4}$—$\frac{1}{64}$	78·3	0·9	2·6	14·7	19·1	73·9	100·0	15·2
	"K"	>1—$\frac{1}{64}$	76·6	2·0	9·6	15·1	13·8	67·2	100·0	12·2
	Average	—	69·5	2·5	9·9	14·5	20·6	70·7	100·0	16·0
DERBYSHIRE	Top Hard	$\frac{1}{2}$—$\frac{1}{64}$	85·4	2·1	4·3	18·7	10·3	69·4	100·0	9·7
	Deep Hard	$\frac{1}{2}$—$\frac{1}{64}$	81·7	3·6	10·8	16·1	7·5	65·2	100·0	9·6
	Waterloo	$\frac{1}{2}$—$\frac{1}{64}$	80·0	1·6	8·1	15·4	11·9	73·1	100·0	11·2
	Average	—	82·4	2·4	7·7	16·7	9·9	69·2	100·0	10·2
NOTTINGHAMSHIRE	"O"	$>\frac{1}{2}$—$\frac{1}{64}$	77·9	3·4	11·9	16·3	10·2	72·6	100·0	12·0
	"D"	$\frac{3}{16}$—$\frac{1}{64}$	44·8	3·8	11·7	16·2	43·5	75·7	100·0	36·5

LANCASHIRE . .	Upper Mountain Mine.	$\frac{1}{2}$—$\frac{1}{64}$	68·1	3·3	22·8	20·9	9·1	55·5	100·0	12·1
	Wigan, 7 ft.	$>\frac{1}{2}$—$\frac{1}{64}$	83·2*	2·2	1·0	30·9	15·8	75·6	100·0	14·1
	Wigan, 4 ft.	$>\frac{1}{2}$—$\frac{1}{64}$	79·2	4·6	2·1	27·2	18·7	73·6	100·0	18·0
	Potato Delf	$\frac{1}{2}$—$\frac{1}{32}$	70·2	2·4	9·2	15·9	20·6	66·0	100·0	16·7
	Higher and Lower Florida.	$\frac{1}{2}$—$\frac{1}{32}$	82·8	1·4	3·2	14·2	14·0	68·9	100·0	11·3
	Average	—	76·7	2·8	7·7	21·8	15·6	67·9	100·0	14·4
CUMBERLAND . .	"R"	$>\frac{1}{8}$—$\frac{1}{64}$	46·2	3·5	5·6	13·7	48·2	85·3	100·0	43·4
	"O"	$\frac{1}{10}$—$\frac{1}{20}$	71·4	3·3	10·6	14·4	18·0	71·7	100·0	16·6
	"H"	$>\frac{3}{8}$—0	43·4	4·1	21·9	16·4	34·7	61·2	100·0	26·6
N. STAFFORDSHIRE .	"S"	$\frac{1}{2}$—$\frac{1}{64}$	59·9	1·9	9·6	16·0	30·5	78·3	100·0	26·6
S. STAFFORDSHIRE .	"B"	$\frac{3}{4}$—$\frac{1}{64}$	52·9	3·8	12·6	19·5	34·5	69·5	100·0	28·5
	"R"	1—$\frac{1}{64}$	75·5	2·6	11·6	16·4	12·9	65·9	100·0	12·4
WARWICKSHIRE . .	"B"	$>\frac{1}{2}$—$\frac{1}{64}$	84·5	1·2	6·7	17·6	8·8	67·6	100·0	8·1
LEICESTERSHIRE . .	"R"	$1\frac{1}{2}$—$\frac{1}{16}$	80·0	2·9	1·6	28·9	18·4	73·0	100·0	16·2
GLOUCESTERSHIRE .	"B"	$\frac{1}{2}$—$\frac{1}{64}$	53·6	3·7	20·4	12·9	26·0	70·3	100·0	22·9
KENT	"A"	$>\frac{1}{4}$—$\frac{1}{64}$	64·7	3·5	18·5	18·7	16·8	61·8	100·0	16·1
	"B"	1—$\frac{1}{64}$	72·3	2·4	13·4	13·1	14·3	73·8	100·0	14·6

* < 1·50 S.G.

Coalfield.	Coal.	Size (in.).	"Pure Coal." < 1·35 S.G. Weight.	Ash.	Middlings. 1·35—1·60 S.G. Weight.	Ash.	Dirt. > 1·60 S.G. Weight.	Ash.	Total. Weight.	Ash.
SCOTLAND . . .	"P"	$\frac{1}{2}$—$\frac{1}{64}$	59·7	3·5	17·0	19·7	23·3	68·8	100·0	21·5
	"K"	1—$\frac{1}{64}$	75·7	3·0	8·0	16·2	16·3	75·4	100·0	17·9
	"R"	$>\frac{1}{2}$—$\frac{1}{64}$	69·8*	5·4	14·4	16·6	15·8	66·2	100·0	16·6
	"W"	$>\frac{1}{2}$—$\frac{1}{64}$	69·4	4·1	8·7	19·3	21·9	74·4	100·0	20·8
	"L"	$>\frac{1}{16}$—$\frac{1}{64}$	52·6	3·3	18·5	15·2	28·9	68·6	100·0	24·3
	Average	—	65·4	3·8	13·3	17·4	21·2	70·5	100·0	20·2
SOUTH WALES . .	"W"	$\frac{1}{2}$—$\frac{1}{64}$	69·5	2·7	25·2	12·4	5·3	71·2	100·0	8·8
	"T"	$1\frac{1}{8}$—$\frac{1}{64}$	65·2	2·1	16·9	11·7	17·9	72·3	100·0	16·3
	"B"	$>\frac{1}{2}$—$\frac{1}{64}$	74·1	2·0	7·9	14·4	18·0	68·8	100·0	15·0
	"M"	$1\frac{1}{2}$—$\frac{1}{4}$	64·2*	3·3	16·8	17·3	19·0	59·2	100·0	17·1
	Anthracite (1)	$\frac{1}{4}$—$\frac{1}{64}$	76·2*	1·3	4·5	12·6	19·3	66·9	100·0	14·5
	Anthracite (2)	$\frac{1}{8}$—$\frac{1}{64}$	85·4†	2·3	1·9	17·4	12·7	80·5	100·0	12·5
	Average	—	72·4	2·3	12·2	14·3	15·4	69·8	100·0	14·0
SUMMARY FOR GREAT BRITAIN—	No. of coals.									
ENGLAND . . .	46	—	72·4	2·7	9·0	19·0	18·6	69·7	100·0	16·3
SCOTLAND . . .	5	—	65·4	3·8	13·3	17·4	21·8	70·5	100·0	20·2
SOUTH WALES . .	6	—	72·4	2·3	12·2	14·3	15·4	69·8	100·0	14·0
GREAT BRITAIN . .	57	—	71·8	2·8	9·7	18·4	18·5	69·7	100·0	16·4

* < 1·40 S.G. † < 1·50 S.G.

Table III.—Float and Sink Results. Sized Samples of Foreign and Dominion Coals

Coal.	Size (in.).	Per cent. of size.	Per cent. by weight and per cent. ash in fractions at different specific gravities.													
			< 1·30 S.G. Weight.	Ash.	1·30—1·40 S.G. Weight.	Ash.	1·40—1·50 S.G. Weight.	Ash.	1·50—1·60 S.G. Weight.	Ash.	1·60—1·70 S.G. Weight.	Ash.	> 1·70 S.G. Weight.	Ash.	Total. Weight.	Ash.
Belgium (1)	2—5/16	40·0	43·0	3·6	12·0	11·7	5·0	23·0	4·0	34·0	3·0	43·3	33·0	75·0	100·0	32·1
	5/16—1/50	49·3	52·5	3·5	10·0	13·3	5·5	25·6	3·5	36·2	2·5	45·0	26·0	71·8	100·0	25·5
	1/50—0	10·7	—	—	—	—	—	—	—	—	—	—	—	—	100·0	22·5
	2—0	100·0	47·7	3·5	11·0	12·4	5·3	24·4	3·7	35·1	2·8	44·2	29·5	73·7	100·0	28·1
(2)	2 1/8—3/4	21·7	43·0	3·5	8·5	14·5	2·5	25·5	1·5	35·2	0·5	42·2	44·0	85·3	100·0	39·8
	3/4—5/16	19·0	48·5	3·2	10·0	19·8	6·0	25·8	3·0	35·0	2·0	46·2	30·5	77·8	100·0	29·3
	5/16—1/50	43·5	50·5	1·8	8·5	12·5	6·5	22·5	3·5	34·2	3·5	42·0	27·5	77·2	100·0	28·7
	1/50—0	15·8	25·0	2·0	23·5	5·0	13·5	15·7	5·0	30·4	6·5	40·7	26·5	70·0	100·0	26·8
	2 1/8—0	100·0	41·8	2·7	12·6	10·8	7·1	20·2	3·3	33·0	3·1	43·2	32·1	78 6	100·0	30·9
(3)	2 9/16—1 1/8	—	18·5	3·5	30·5	9·0	7·5	14·5	3·0	21·0	4·0	28·5	36·5	61·0	100·0	28·5
	1 1/8—3/4	—	6·5	2·5	35·0	5·0	9·0	15·0	6·0	22·0	4·5	28·0	39·0	60·0	100·0	30·7
	3/4—1/16	—	13·5	2·0	31·5	5·5	7·5	15·5	5·5	22·5	4·0	31·0	38·0	64·0	100·0	30·0
	5/16—1/50	—	15·5	1·5	33·5	4·5	4·0	15·0	7·0	24·5	5·0	35·0	35·0	67·0	100·0	29·2
	1/50—1/125	—	27·5	1·0	24·5	5·2	6·5	16·0	3·5	26·0	4·5	36·5	33·5	70·0	100·0	28·6
	2 9/16—1/125	—	16·3	1·9	31·0	5 3	6·9	15·2	5·0	23·6	4·4	32·0	36·4	64·3	100·0	29·4
(4)	2—1 3/16	12·0	36·0	3·2	31·0	6·0	2·5	16·8	1·5	27·5	0·5	35·0	28·5	76·0	100·0	26·0
	1 3/16—3/4	11·9	32·5	3·2	26·5	6·0	2·5	19·5	2·5	28·0	1·0	38·0	35·0	75·7	100·0	30·5
	3/4—3/16	25·9	18·5	2·1	33·0	5·3	3·0	18·5	2·5	31·5	2·0	45·8	41·0	76·8	100·0	35·9
	3/16—1/50	36·9	49·3	2·7	13·0	9·8	3·0	21·5	2·5	30·0	1·5	45·0	30·5	74·5	100·0	27·4
	1/50—0	13·3	—	—	—	—	—	—	—	—	—	—	—	—	100·0	24·2
	2—0	100·0	34·0	2·9	25·9	6·3	2·7	19·1	2·3	29·4	1·3	42·9	33·8	79·3	100·0	29·0

Coal.	Size (in.).	Per cent. of size.	< 1·30 S.G. Weight.	< 1·30 S.G. Ash.	1·30—1·40 S.G. Weight.	1·30—1·40 S.G. Ash.	1·40—1·50 S.G. Weight.	1·40—1·50 S.G. Ash.	1·50—1·60 S.G. Weight.	1·50—1·60 S.G. Ash.	1·60—1·70 S.G. Weight.	1·60—1·70 S.G. Ash.	> 1·70 S.G. Weight.	> 1·70 S.G. Ash.	Total. Weight.	Total. Ash.
			Per cent. by weight and per cent. ash in fractions at different specific gravities.													
(5) .	2—$1\frac{3}{16}$	—	44·5	3·0	2·0	10·0	2·5	18·5	2·0	28·0	3·0	37·0	46·0	78·0	100·0	33·1
	$1\frac{3}{16}$—$\frac{3}{4}$	—	50·5	3·0	6·0	10·0	2·5	20·0	1·5	29·0	1·5	36·5	38·0	77·5	100·0	22·4
	$\frac{3}{4}$—$\frac{3}{8}$	—	63·0	3·5	5·5	11·0	2·0	20·0	2·5	27·5	2·5	36·5	24·5	77·0	100·0	18·6
	$\frac{3}{8}$—$\frac{3}{16}$	—	66·5	4·0	8·0	10·7	3·0	20·5	3·0	29·0	1·5	38·0	18·0	76·5	100·0	18·6
	$\frac{3}{16}$—$\frac{1}{50}$	—	74·0	3·5	3·5	17·5	2·0	25·0	1·5	32·0	1·0	37·0	18·0	72·5	100·0	17·4
	$\frac{1}{50}$—0	—	—	—	—	—	—	—	—	—	—	—	—	—	100·0	11·8
	2—0	—	59·7	3·5	5·0	11·5	2·4	20·6	2·1	28·9	1·9	36·9	28·9	76·8	100·0	—
(6) .	2—$\frac{3}{8}$	41·0	9·0	1·5	51·0	5·2	7·0	20·0	5·0	31·0	4·0	40·0	24·0	66·0	100·0	23·2
	$\frac{3}{8}$—$\frac{1}{50}$	50·1	40·5	3·0	19·5	9·5	6·0	22·5	5·5	31·0	2·5	38·5	26·0	64·0	100·0	23·8
	$\frac{1}{50}$—0	8·9	—	—	—	—	—	—	—	—	—	—	—	—	100·0	26·6
	2—0	100·0	24·7	2·7	35·2	6·4	6·5	21·1	5·3	31·0	3·3	39·5	25·0	64·9	100·0	23·8
(7) .	$2\frac{3}{4}$—$\frac{5}{16}$	—	41·0	3·0	9·5	10·3	4·0	24·3	2·0	32·0	1·0	35·7	42·5	74·5	100·0	35·8
	$\frac{5}{16}$—$\frac{1}{50}$	—	50·0	2·5	7·0	11·3	8·0	22·0	3·5	32·5	3·0	40·0	28·5	73·0	100·0	26·9
	$2\frac{3}{4}$—$\frac{1}{50}$	—	45·6	2·7	8·2	10·7	6·0	22·7	2·7	32·4	2·0	38·9	35·5	73·8	100·0	—
(8) .	$2\frac{3}{4}$—$\frac{5}{16}$	—	39·0	2·7	5·5	14·7	1·5	23·5	1·5	33·0	1·5	42·5	51·0	78·5	100·0	43·3
	$\frac{5}{16}$—$\frac{1}{50}$	—	47·5	2·6	5·0	10·7	4·0	20·5	1·5	29·5	2·0	36·5	40·0	78·0	100·0	35·3
	$2\frac{3}{4}$—$\frac{1}{50}$	—	43·3	2·6	5·2	12·8	2·7	21·2	1·5	31·3	1·8	39·1	45·5	78·4	100·0	—
(9) .	2—$\frac{3}{8}$	—	14·0	3·0	20·5	6·0	7·5	15·0	5·0	23·5	3·5	34·0	49·5	71·0	100·0	40·6
	$\frac{3}{8}$—$\frac{1}{50}$	—	26·0	2·5	28·0	7·0	9·0	16·4	6·0	23·5	4·5	34·5	26·5	69·0	100·0	25·4
	$\frac{1}{50}$—0	—	—	—	—	—	—	—	—	—	—	—	—	—	100·0	20·0
	2—0	—	20·0	2·6	24·2	6·6	8·3	15·8	5·5	23·5	4·0	34·3	38·0	70·3	100·0	—

(10)	$\frac{3}{8}$—$\frac{1}{50}$	—	52·5	2·0	19·0	6·8	6·5	15·0	3·0	27·0	2·0	35·0	17·0	68·7	100·0	16·5
FRANCE (1).	$\frac{5}{16}$—$\frac{1}{25}$	62·8	37·5	2·5	13·5	10·7	7·0	22·5	5·0	32·0	5·0	40·0	32·0	70·2	100·0	29·8
	$\frac{1}{25}$—$\frac{1}{50}$	13·6	37·5	2·5	22·5	7·0	5·0	20·2	4·5	29·2	3·5	40·0	27·0	70·4	100·0	29·7
	$\frac{1}{50}$—0	23·6	—	—	—	—	—	—	—	—	—	—	—	—	100·0	24·8
	$\frac{5}{16}$—0	100·0	37·5	2·5	18·0	8·4	6·0	21·5	4·8	30·7	4·2	40·0	29·5	70·3	100·0	28·6
(2).	2—1$\frac{3}{16}$	—	13·5	4·2	34·0	7·7	4·0	23·5	4·5	33·0	2·5	38·0	41·5	55·0	100·0	29·6
SARRE (1).	$\frac{5}{16}$—$\frac{5}{32}$	15·3	54·0	4·6	4·5	14·3	3·5	23·7	1·5	31·5	2·0	41·5	34·5	75·7	100·0	36·0
	$\frac{5}{32}$—$\frac{1}{50}$	55·1	31·0	3·0	15·0	9·6	5·0	15·0	3·0	25·0	3·5	36·3	42·5	74·8	100·0	37·5
	$\frac{1}{50}$—0	29·6	—	—	—	—	—	—	—	—	—	—	—	—	100·0	31·6
	$\frac{5}{16}$—0	100·0	42·5	4·0	9·8	10·7	4·2	18·6	2·3	27·8	2·7	38·2	38·5	75·2	100·0	35·5
GERMANY (1).	2$\frac{3}{8}$—$\frac{3}{8}$	46·2	33·0	2·0	20·0	9·0	4·0	20·5	2·5	36·0	0·5	40·0	40·0	81·5	100·0	37·5
	$\frac{3}{8}$—$\frac{1}{50}$	40·0	34·5	0·5	25·5	4·5	6·5	18·0	4·0	25·5	2·0	46·5	27·5	73·0	100·0	25·0
	$\frac{1}{50}$—0	13·8	—	—	—	—	—	—	—	—	—	—	—	—	100·0	21·0
	2$\frac{3}{8}$—0	100·0	33·7	1·2	22·8	6·5	5·3	18·9	3·2	29·6	1·2	45·4	33·8	79·2	100·0	30·2
(2).	1$\frac{3}{8}$—$\frac{3}{8}$	36·7	55·0	3·0	7·5	17·0	3·0	27·0	2·0	33·0	1·5	38·0	31·0	71·0	100·0	27·0
	$\frac{3}{8}$—$\frac{1}{50}$	47·5	62·5	2·0	4·5	14·0	3·0	23·0	1·0	29·0	3·0	38·0	26·0	72·0	100·0	23·5
	$\frac{1}{50}$—0	15·8	—	—	—	—	—	—	—	—	—	—	—	—	100·0	27·0
	1$\frac{3}{8}$—0	100·0	58·8	2·8	0·0	16·5	3·0	25·0	1·5	31·7	2·2	38·0	28·5	71·5	100·0	25·3
(3).	1$\frac{1}{16}$—$\frac{3}{8}$	—	52·5	2·0	25·0	8·0	2·0	21·0	1·5	27·0	1·0	35·0	18·0	76·0	100·0	18·0
	$\frac{3}{8}$—$\frac{1}{50}$	—	67·0	2·5	13·0	9·0	2·0	22·0	1·0	30·0	1·0	37·0	16·0	74·0	100·0	15·8
	$\frac{1}{50}$—0	—	—	—	—	—	—	—	—	—	—	—	—	—	100·0	13·3
	1$\frac{3}{16}$—0	—	59·8	2·3	19·0	8·3	2·0	21·5	1·2	28·2	1·0	36·0	17·0	75·0	100·0	—

Coal.	Size (in.).	Per cent. of size.	Per cent. by weight and per cent. ash in fractions at different specific gravities.													
			< 1·30 S.G.		1·30—1·40 S.G.		1·40—1·50 S.G.		1·50—1·60 S.G.		1·60—1·70 S.G.		> 1·70 S.G.		Total.	
			Weight.	Ash.	Weight.	Ash.	Weight.	Ash.	Weight.	Ash.	Weight.	Ash.	Weight.	Ash.	Weight.	Ash.
(4) .	$\frac{3}{32}$—$\frac{1}{50}$	—	37·5	3·6	26·5	11·0	6·0	23·3	3·5	33·1	2·5	42·2	24·0	78·4	100·0	26·7
	$\frac{1}{50}$—$\frac{1}{150}$	—	35·3	3·5	18·0	10·5	6·7	20·9	3·8	31·0	3·2	40·4	33·0	73·5	100·0	31·4
	$\frac{3}{32}$—$\frac{1}{150}$	—	36·4	3·6	22·3	10·8	6·4	22·0	3·6	32·1	2·8	41·1	28·5	75·4	100·0	—
(5) .	$\frac{3}{8}$—$\frac{1}{50}$	75·7	70·2	2·5	7·3	9·7	2·7	20·0	1·8	29·0	1·2	36·0	16·8	68·5	100·0	14·0
	$\frac{1}{50}$—0	24·3	—	—	—	—	—	—	—	—	—	—	—	—	100·0	14·6
(6) .	$\frac{3}{8}$—$\frac{1}{50}$	75·8	47·0	1·5	23·2	4·5	5·8	14·0	2·5	23·8	2·0	33·7	19·5	70·5	100·0	17·1
	$\frac{1}{50}$—0	24.2	—	—	—	—	—	—	—	—	—	—	—	—	100·0	15·2
(7) .	$\frac{3}{8}$—$\frac{1}{50}$	76·6	70·5	1·1	6·2	9·0	2·6	16·7	1·7	24·5	1·5	33·3	17·5	70·7	100·0	15·6
	$\frac{1}{50}$—0	23·4	—	—	—	—	—	—	—	—	—	—	—	—	100·0	13·3
(8) .	$\frac{3}{8}$—$\frac{1}{50}$	70·4	79·5	2·5	2·5	15·0	1·5	27·5	1·0	36·5	1·0	43·0	14·5	73·5	100·0	15·0
	$\frac{1}{50}$—0	29·6	—	—	—	—	—	—	—	—	—	—	—	—	100·0	18·4
(9) .	$\frac{3}{8}$—$\frac{1}{50}$	75·6	61·0	3·5	8·0	11·0	5·0	21·0	2·0	33·5	2·0	42·0	22·0	78·0	100·0	22·2
	$\frac{1}{50}$—0	24·4	—	—	—	—	—	—	—	—	—	—	—	—	100·0	26·0
(10) .	$\frac{3}{8}$—$\frac{1}{50}$	79·5	69·5	3·0	3·5	11·0	3·5	25·5	3·5	36·0	2·0	44·0	18·0	71·5	100·0	18·3
	$\frac{1}{50}$—0	20·5	—	—	—	—	—	—	—	—	—	—	—	—	100·0	19·7
(11) .	$\frac{3}{8}$—$\frac{1}{50}$	—	49·0	4·0	25·0	10·5	13·2	19·5	3·8	31·0	1·0	37·0	8·0	66·9	100·0	18·6
(12) .	$\frac{3}{8}$—$\frac{1}{50}$	—	55·5	2·0	29·5	6·5	4·0	19·5	2·0	25·5	1·5	33·0	6·5	64·8	100·0	8·9

(13) .	3⅛—⅜	—	63·2	2·1	8·7	13·0	2·3	20·5	1·2	30·5	1·5	41·8	11·0	69·9	100·0	11·6
	⅜—1/50	—	80·2	1·8	5·1	8·0	1·7	15·5	1·0	29·0	1·0	43·5	22·5	75·4	100·0	19·7
	3⅛—1/50	—	72·0	1·9	6·9	11·1	2·0	18·4	1·1	29·8	1·2	42·5	16·8	73·2	100·0	—
(14) .	⅜—1/50	—	43·5	2·0	9·5	9·8	14·5	20·0	10·0	31·8	5·7	40·5	16·8	59·4	100·0	19·8
(15) .	2—⅜	—	75·5	3·0	11·0	11·8	2·0	20·5	1·2	28·5	0·8	40·8	9·5	70·0	100·0	11·3
	⅜—1/50	—	84·5	2·5	5·8	12·0	1·2	24·5	0·5	29·2	0·8	36·0	7·2	66·0	100·0	8·3
	1/50—0	—	—	—	—	—	—	—	—	—	—	—	—	—	100·0	9·1
	2—0	—	80·0	2·7	8·4	11·9	1·6	22·0	0·9	28·7	0·8	38·4	8·3	68·3	100·0	—
(16) .	2¾—⅜	—	39·4	2·7	11·5	8·3	6·4	20·0	2·3	30·4	1·3	36·2	39·1	82·0	100·0	36·5
	⅜—1/50	—	59·5	2·2	11·5	9·2	3·4	21·0	1·5	29·5	1·0	36·5	23·1	79·0	100·0	22·1
	1/50—0	—	—	—	—	—	—	—	—	—	—	—	—	—	100·0	21·3
	2¾—0	—	49·5	2·4	11·5	8·7	4·9	20·4	1·9	30·0	1·1	36·3	31·1	80·9	100·0	—
SPAIN (1). .	2—5/16	—	52·5	3·0	18·5	7·5	2·0	16·0	1·0	27·0	1·0	35·0	25·0	80·6	100·0	24·0
	5/16—0	—	42·5	3·5	24·5	7·5	5·5	19·8	2·0	31·0	2·0	40·2	23·5	83·4	100·0	25·4
	2—0	—	47·5	3·2	21·5	7·5	3·8	18·8	1·5	29·7	1·5	38·5	24·2	82·1	100·0	—
(2). .	2—5/16	—	6·0	3·1	31·5	11·0	27·0	22·5	9·0	34·5	5·5	46·0	21·0	66·0	100·0	27·9
	5/16—1/50	—	18·5	6·0	42·0	11·5	9·0	28·5	5·0	32·2	3·5	39·5	22·0	67·5	100·0	26·2
	2—1/50	—	12·3	5·3	36·7	11·3	18·0	24·0	7·0	33·7	4·5	43·5	21·5	67·0	100·0	—
(3). .	2⅛—5/16	—	32·0	7·5	18·5	15·7	8·0	26·0	5·7	36·6	3·3	45·6	32·5	79·0	100·0	36·7
	5/16—1/50	—	36·2	7·0	19·3	16·2	7·5	25·7	6·0	35·0	2·4	43·5	28·6	78·0	100·0	31·8
	1/50—0	—	—	—	—	—	—	—	—	—	—	—	—	—	100·0	38·7
	2⅛—0	—	34·1	7·2	18·9	15·9	7·8	25·8	5·9	35·7	2·8	44·8	30·5	78·4	100·0	—

Coal.	Size (in.).	Per cent. of size.	Per cent. by weight and per cent. ash in fractions at different specific gravities.													
			< 1·30 S.G.		1·30—1·40 S.G.		1·40—1·50 S.G.		1·50—1·60 S.G.		1·60—1·70 S.G.		> 1·70 S.G.		Total.	
			Weight.	Ash.	Weight.	Ash.	Weight.	Ash.	Weight.	Ash.	Weight.	Ash.	Weight.	Ash.	Weight.	Ash.
(4).	$\frac{1}{8}$—1	70·3	41·0	6·0	9·2	18·2	6·8	26·4	6·0	34·0	7·1	43·0	29·9	71·8	100·0	32·0
	$\frac{1}{50}$—0	29·7	—	—	—	—	—	—	—	—	—	—	—	—	100·0	34·4
HOLLAND (1).	$\frac{9}{32}$—$\frac{1}{50}$	72·2	33·0	1·5	46·0	3·5	2·5	16·3	2·0	22·5	1·0	33·0	15·5	71·0	100·0	14·6
	$\frac{1}{50}$—0	27·8	—	—	—	—	—	—	—	—	—	—	—	—	100·0	15·7
SERBIA (1).	$1\frac{3}{16}$—$\frac{3}{16}$	—	3·5	9·0	55·5	12·0	20·0	21·0	7·0	32·0	4·0	43·5	10·0	63·0	100·0	21·4
	$\frac{3}{16}$—$\frac{1}{50}$	—	6·5	10·5	67·0	14·5	15·5	23·5	4·0	32·0	2·0	41·0	5·0	60·0	100·0	19·1
	$1\frac{3}{16}$—$\frac{1}{50}$	—	5·0	10·0	61·2	13·4	17·8	22·1	5·5	32·0	3·0	42·7	7·5	62·0	100·0	—
(2).	$1\frac{3}{16}$—$\frac{3}{16}$	—	—	—	7·5	11·5	28·0	18·2	20·8	27·3	10·2	33·6	33·5	58·5	100·0	34·7
	$\frac{3}{16}$—$\frac{1}{50}$	—	3·7	9·0	27·3	13·2	32·5	21·2	14·5	28·7	6·0	34·3	16·0	55·5	100·0	26·0
	$1\frac{3}{16}$—$\frac{1}{50}$	—	1·9	9·0	17·5	12·9	30·2	19·8	17·6	27·8	8·1	33·8	24·7	57·6	100·0	—
CZECHO-SLOVAKIA (1).	2—$\frac{5}{16}$	—	—	—	30·5	5·5	13·5	23·5	10·0	31·0	3·0	39·5	43·0	72·5	100·0	40·3
	$\frac{5}{16}$—$\frac{1}{50}$	—	29·0	2·5	15·0	9·0	6·5	22·0	4·0	32·5	2·5	39·5	43·0	75·0	100·0	38·0
	$\frac{1}{50}$—0	—	—	—	—	—	—	—	—	—	—	—	—	—	100·0	35·9
	2—0	—	14·5	2·5	22·7	6·8	10·0	23·2	7·0	31·4	2·8	39·5	43·0	74·0	100·0	—

(2).	$\frac{3}{4}$—$\frac{3}{8}$	35·0	7·5	6·0	6·0	15·0	7·0	25·0	6·5	38·5	5·0	40·0	68·0	65·0	100·0	49·7
	$\frac{3}{8}$—$\frac{1}{50}$	48·9	16·5	5·5	10·5	18·0	8·5	28·0	8·0	37·0	6·5	45·0	50·0	68·0	100·0	43·7
	$\frac{1}{50}$—0	16·1	—	—	—	—	—	—	—	—	—	—	—	—	100·0	44·1
	$\frac{3}{4}$—0	100·0	12·0	5·6	8·3	16·9	7·7	26·6	7·3	37·6	5·7	42·8	59·0	66·2	100·0	45·8
AUSTRIA-HUNGARY (1).	$\frac{3}{4}$—$\frac{3}{8}$	—	5·5	4·1	17·0	10·0	18·0	20·0	14·5	30·8	9·5	41·8	35·5	64·8	100·0	37·0
	$\frac{3}{8}$—0	—	9·5	5·6	20·0	10·0	18·0	21·3	12·1	33·1	9·4	46·0	31·0	66·5	100·0	35·3
	$\frac{3}{4}$—0	—	7·5	5·0	18·5	10·0	18·0	20·7	13·3	31·8	9·5	43·9	33·2	65·6	100·0	—
(2).	$\frac{5}{8}$—$\frac{3}{8}$	—	32·0	2·1	25·8	6·4	6·1	16·8	4·1	28·2	3·0	40·5	29·0	75·0	100·0	26·8
	$\frac{3}{8}$—$\frac{1}{50}$	—	65·5	2·3	12·7	6·8	4·8	18·9	2·5	29·8	1·5	39·0	13·0	71·8	100·0	14·0
	$\frac{1}{50}$—0	—	—	—	—	—	—	—	—	—	—	—	—	—	100·0	13·9
	$\frac{5}{8}$—0	—	48·7	2·2	19·3	6·5	5·4	17·8	3·3	28·8	2·3	40·0	21·0	74·0	100·0	—
(3).	2—$\frac{3}{8}$	68·8	2·0	6·5	17·1	14·0	30·1	24·5	19·6	36·5	19·2	44·5	12·0	63·2	100·0	33·2
	$\frac{3}{8}$—$\frac{1}{50}$	25·5	11·0	4·0	25·0	11·0	21·5	22·5	11·0	32·5	10·0	42·5	21·5	63·4	100·0	29·5
	$\frac{1}{50}$—0	5·7	—	—	19·0	8·0	13·6	17·5	9·9	27·4	13·5	40·5	44·0	61·7	100·0	39·0
	2—0	100·0	4·3	4·4	20·4	10·9	21·7	22·0	12·5	33·2	14·2	42·7	25·9	62·4	100·0	32·5
POLAND (1).	>$\frac{1}{125}$	75·7	48·9	4·2	19·4	11·4	5·6	19·4	9·7	27·1	4·4	36·5	12·0	61·6	100·0	17·1
	$\frac{1}{125}$—0	24·3	—	—	—	—	—	—	—	—	—	—	—	—	100·0	29·8

Coal.	Size (in.).	Per cent. of size.	Per cent. by weight and per cent ash in fractions at different specific gravities.													
			< 1·30 S.G.		1·30—1·40 S.G.		1·40—1·50 S.G.		1·50—1·60 S.G.		1·60—1·70 S.G.		> 1·70 S.G.		Total.	
			Weight.	Ash.	Weight.	Ash.	Weight.	Ash.	Weight.	Ash.	Weight.	Ash.	Weight.	Ash.	Weight.	Ash.
RUSSIA (1).	$2\frac{3}{8}$—$\frac{3}{8}$	40·0	22·0	2·8	56·0	6·0	5·0	19·0	2·5	32·0	1·0	39·2	15·5	72·0	100·0	17·0
	$\frac{3}{8}$—$\frac{1}{50}$	43·0	14·0	1·8	54·0	2·8	4·7	12·5	2·3	26·0	1·5	32·7	23·5	66·5	100·0	19·0
	$\frac{1}{50}$—0	17·0	—	—	—	—	—	—	—	—	—	—	—	—	100·0	60·0
	$2\frac{3}{8}$—0	100·0	18·0	2·4	55·0	4·4	3·8	14·9	2·4	29·2	1·3	35·3	19·5	68·5	100·0	25·2
TURKEY (1).	2—$\frac{3}{8}$	28·8	26·8	5·5	20·7	9·5	9·5	19·5	3·5	34·7	2·8	39·3	36·7	80·3	100·0	37·9
	$\frac{3}{8}$—$\frac{1}{50}$	58·5	33·5	4·3	24·7	9·0	9·8	18·3	5·1	29·8	2·4	41·0	24·5	75·5	100·0	26·5
	$\frac{1}{50}$—0	12·7	—	—	—	—	—	—	—	—	—	—	—	—	100·0	25·0
	2—0	100·0	30·1	4·8	22·7	9·2	9·7	18·8	4·3	31·8	2·6	40·1	30·6	79·3	100·0	29·6
CHINA (1).	$\frac{5}{8}$—$\frac{3}{8}$	32·4	79·5	6·2	5·5	16·1	3·0	26·3	1·8	36·5	1·2	47·0	9·0	69·5	100·0	13·4
	$\frac{3}{8}$—$\frac{1}{50}$	59·0	80·8	3·5	4·0	16·2	2·7	24·5	2·2	33·7	1·5	41·8	8·8	75·0	100·0	12·1
	$\frac{1}{50}$—0	8·6	—	—	—	—	—	—	—	—	—	—	—	—	100·0	24·5
	$\frac{5}{8}$—0	100·0	80·1	4·8	4·8	16·1	2·8	25·6	2·0	34·9	1·4	44·2	8·9	72·2	100·0	13·4

Coal.	Size (in).	Per cent. of size.	Per cent. by weight and per cent. ash in fractions at different specific gravities.											
			< 1·35 S.G. Weight.	Ash.	1·35—1·40 S.G. Weight.	Ash.	1·40—1·50 S.G. Weight.	Ash.	1·50—1·60 S.G. Weight.	Ash.	> 1·60 S.G. Weight.	Ash.	Total. Weight.	Ash.
SOUTH AFRICA "A" .	1—¼	100·0	16·0	6·4	27·4	7·8	35·0	12·9	11·7	22·7	9·9	42·7	100·0	14·6
"B" .	2—½	100·0	22·4	6·5	21·8	8·8	32·6	13·4	8·0	21·8	15·2	43·6	100·0	16·1
AUSTRALIA "A". .	>⅜	28·5	55·7	6·8	14·0	13·9	7·2	20·2	3·7	32·7	19·4	67·5	100·0	21·5
	⅜—⅛	40·7	59·1	5·8	16·9	12·8	7·4	19·7	2·5	31·1	14·1	72·4	100·0	18·1
	⅛—1/32	19·6	71·1	5·0	4·9	11·7	7·4	19·7	1·9	28·9	14·7	72·1	100·0	16·7
	1/32—0	11·2	53·9	5·6	12·5	10·7	10·6	18·6	2·6	28·2	20·4	65·4	100·0	20·3
	>⅜—0	100·0	60·0	5·7	12·[illegible]	12·5	8·1	19·5	2·7	31·3	17·1	68·9	100·0	19·0
"B". .	>⅜	22·2	55·6	6·6	18·0	12·0	10·0	19·4	1·2	34·3	15·2	61·9	100·0	17·6
	⅜—⅛	37·6	70·8	6·1	12·2	13·5	6·6	19·2	1·6	27·7	8·8	64·1	100·0	13·3
	⅛—1/32	25·2	85·8	5·2	2·2	13·9	3·9	19·1	1·8	27·7	6·3	62·6	100·0	9·9
	1/32—0	15·0	91·8	7·7	—	—	1·8	14·5	—	—	6·4	60·6	100·0	11·2
	>⅜—0	100·0	76·0	6·4	8 [illegible]	12·7	5·6	18·9	1·1	29·4	9·2	62·4	100·0	13·1
INDIA "A". .	2—1	58·7	5·6	8·0	18·2	10·6	40·6	14·9	8·4	25·1	27·2	46·4	100·0	23·2
	1—½	41·3	5·9	7·4	15·2	10·7	37·7	14·6	10·3	22·5	30·9	47·9	100·0	24·7
	2—½	100·0	5·8	7·7	16·7	10·6	39·1	14·8	9·3	23·8	29·1	47·1	100·0	23·8

TABLE IV.—SUMMARY OF FLOAT AND SINK RESULTS. FOREIGN AND DOMINION COALS

Coalfield.	Coal.	Size (in.).	"Pure Coal." < 1·30 S.G. Weight.	Ash.	Middlings. 1·30—1·60 S.G. Weight.	Ash.	Dirt. > 1·60 S.G. Weight.	Ash.	Total. Weight.	Ash.
BELGIUM	(1)	2 —1/50	47·7	3·5	20·0	19·8	32·3	71·2	100·0	28·6
	(2)	2 1/8 —0	41·8	2·7	23·0	16·9	35·2	75·3	100·0	31·5
	(3)	2 3/16—1/50	16·3	1·9	42·9	9·4	40·8	60·8	100·0	31·9
	(4)	2 —1/50	34·0	2·9	30·9	9·2	35·1	77·9	100·0	31·2
	(5)	2 —1/50	59·7	3·5	9·5	17·7	30·8	74·4	100·0	26·7
	(6)	2 —1/50	24·7	2·7	47·0	11·2	28·3	62·2	100·0	23·5
	(7)	2 3/4 —1/50	45·6	2·7	16·9	18·4	37·5	71·6	100·0	31·1
	(8)	2 3/4 —1/50	43·3	2·6	9·4	18·2	47·3	77·1	100·0	39·2
	(9)	2 —1/50	20·0	2·6	38·0	11·0	42·0	66·9	100·0	32·8
	(10)	3/8 —1/50	52·5	2·0	28·5	10·8	19·0	65·2	100·0	16·5
	Average	—	38·6	2·7	26·6	14·3	34·8	70·2	100·0	29·2
FRANCE	(1)	5/16—1/50	37·5	2·5	28·8	14·8	33·8	66·4	100·0	27·6
	(2)	2 —1 3/16	13·5	4·2	42·5	11·9	44·0	54·6	100·0	29·6
SARRE	(1)	5/16—1/50	42·5	4·0	16·3	18·9	41·2	72·6	100·0	34·6
GERMANY	(1)	2 3/8 —1/50	33·7	1·2	31·3	11·0	35·0	78·0	100·0	31·1
	(2)	1 3/8 —1/50	55·8	2·8	10·5	21·1	30·7	69·0	100·0	25·1
	(3)	1 3/16—1/50	59·8	2·3	22·2	10·7	18·0	72·8	100·0	16·9
	(4)	5/32—1/150	36·4	3·6	32·3	15·4	31·3	72·0	100·0	28·8
	(5)	3/8 —1/50	70·2	2·5	11·8	15·0	18·0	66·3	100·0	15·5
	(6)	3/8 —1/50	47·0	1·5	31·5	8·1	21·5	67·2	100·0	17·6
	(7)	3/8 —1/50	70·5	1·1	10·5	13·4	19·0	67·8	100·0	15·0

	(8)	$\frac{3}{8}$—$\frac{1}{50}$	79·5	2·5	5·0	23·1	15·5	71·5	100·0	14·2
	(9)	$\frac{3}{8}$—$\frac{1}{50}$	61·0	3·5	15·0	17·3	24·0	75·3	100·0	22·8
	(10)	$\frac{3}{8}$—$\frac{1}{50}$	69·5	3·0	10·5	24·0	20·0	68·7	100·0	18·3
	(11)	$\frac{3}{8}$—$\frac{1}{50}$	74·0*	6·2	17·0	20·4	9·0	63·7	100·0	13·8
	(12)	$\frac{3}{8}$—$\frac{1}{50}$	85·0*	3·6	6·0	21·5	8·0	57·6	100·0	8·9
	(13)	$3\frac{1}{8}$—$\frac{1}{50}$	78·9	2·8	3·1	22·4	18·0	71·1	100·0	15·6
	(14)	$\frac{3}{8}$—$\frac{1}{50}$	53·0	3·4	24·5	23·8	22·5	54·7	100·0	19·8
	(15)	2—$\frac{1}{50}$	80·0	2·7	10·9	14·7	9·1	65·7	100·0	9·7
	(16)	$2\frac{3}{4}$—$\frac{1}{50}$	49·5	2·4	18·3	14·0	32·2	79·3	100·0	29·4
	Average .	—	63·0	2·8	16·3	17·2	20·7	69·2	100·0	18·9
SPAIN	(1)	2—0	47·5	3·2	26·8	10·4	25·7	79·3	100·0	24·8
	(2)	2—$\frac{1}{50}$	12·3	5·3	61·7	19·9	26·0	62·9	100·0	29·3
	(3)	$2\frac{1}{8}$—$\frac{1}{50}$	34·1	7·2	32·6	21·8	33·3	75·5	100·0	34·6
	(4)	$\frac{3}{8}$—$\frac{1}{50}$	41·0	6·0	22·0	25·0	37·0	66·3	100·0	32·4
	Average .	—	33·7	5·4	35·8	19·3	30·5	71·0	100·0	30·3
HOLLAND	(1)	$\frac{9}{32}$—$\frac{1}{50}$	79·0*	2·7	4·5	19·1	16·5	68·8	100·0	14·3
SERBIA	(1)	$1\frac{3}{16}$—$\frac{1}{50}$	5·0	10·0	84·3	16·5	10·5	56·4	100·0	20·0
	(2)	2—$\frac{1}{50}$	1·9	9·0	65·3	20·1	32·8	51·8	100·0	30·2
CZECHO-SLOVAKIA . .	(1)	2—0	14·5	2·5	39·7	16·6	45·8	71·8	100·0	39·9
	(2)	$\frac{3}{4}$—$\frac{1}{50}$	12·0	5·6	23·4	26·6	64·8	63·8	100·0	48·3
AUSTRIA-HUNGARY . .	(1)	$\frac{3}{4}$—0	7·5	5·0	49·8	19·7	42·7	60·8	100·0	36·1
	(2)	$\frac{5}{8}$—0	48·8	2·2	28·0	10·2	23·2	68·7	100·0	19·8
	(3)	2—$\frac{1}{50}$	4·3	4·4	55·6	20·6	40·1	55·3	100·0	33·7

* < 1·40 S.G.

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